NORTH OF ENGLAND INSTITUTE

OF

MINING AND MECHANICAL ENGINEERS.

DIRECTIONS TO BINDER.

VOLUME XLIII.

OF THE

TRANSACTIONS

COMPRISES

Parts 1, 2, 3, 4, 5, and 6, and ANNUAL REPORT.

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NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

TRANSACTIONS

VOL. XLIII.

1893–94.

EDITED BY M. WALTON BROWN, SECRETARY.

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NORTH OF ENGLAND INSTITUTE

OF

MINING AND MECHANICAL ENGINEERS.

ANNUAL GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
AUGUST 5TH, 1893.

MR. J. B. SIMPSON, RETIRING PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings on July 29th and that day. The Secretary also reported the proceedings of the Council of the Federated Institution of Mining Engineers.

ELECTION OF OFFICERS.

The CHAIRMAN appointed Messrs. R. A. S. Redmayne, J. D. Wilson, John Wood, W. E. Nicholson, L. Thursby, and R. Dodd as scrutineers of the balloting papers for the election of officers for the ensuing year. These gentlemen afterwards reported the result of the ballot as follows :—

PRESIDENT.

Mr. A. L. STEAVENSON, Durham.

VICE-PRESIDENTS.

Mr. T. W. BENSON, 11, Newgate Street, Newcastle-upon-Tyne.
Mr. C. BERKLEY, Marley Hill, Swalwell, R.S.O., Co. Durham.
Mr. T. DOUGLAS, The Garth, Darlington.
Mr. W. F. HALL, Haswell Colliery, Haswell, *via* Sunderland.
Mr. J. L. HEDLEY, 22, Hawthorn Terrace, Newcastle-upon-Tyne.
Mr. G. MAY, Harton Colliery, near South Shields.

COUNCIL.

Mr. H. ARMSTRONG, Chester-le-Street.
Mr. W. C. BLACKETT, Acorn Close, Sacriston, Durham.
Mr. V. W. CORBETT, Chilton Moor, Fence Houses.
Mr. T. E. FORSTER, 3, Eldon Square, Newcastle-upon-Tyne.
Mr. W. H. HEDLEY, Medomsley, R.S.O., Newcastle-upon-Tyne.
Mr. T. HEPPELL, Leafield House, Chester-le-Street.
Mr. H. LAWRENCE, Grange Iron Works, Durham.
Mr. C. C. LEACH, Seghill Colliery, Northumberland.

Mr. T. Lishman, Hetton Colliery, Hetton-le-Hole, R.S.O.
Mr. H. Palmer, East Howle Colliery, near Ferryhill.
Mr. M. W. Parrington, Wearmouth Colliery, Sunderland.
Mr. A. M. Potter, Riding Mill-upon-Tyne.
Mr. R. Robinson, Howlish Hall, near Bishop Auckland.
Mr. T. O. Robson, Chowdene Cottage, Gateshead-upon-Tyne.
Mr. T. H. M. Stratton, Cramlington House, Northumberland.
Mr. S. Tate, Trimdon Grange Colliery, Co. Durham.
Mr. R. L. Weeks, Willington, Co. Durham.
Mr. W. O. Wood, South Hetton, Sunderland.

Ex-Officio

Sir George Elliot, Bart., 1, Park Street, Park Lane, London, W.
Lord Armstrong, C.B., Cragside, Rothbury.
Mr. Lindsay Wood, The Hermitage, Chester-le-Street.
Mr. G. B. Forster, 3, Eldon Square, Newcastle-upon-Tyne.
Mr. John Daglish, Rothley Lake, Cambo, R.S.O., Northumberland.
Sir Lowthian Bell, Bart., Rounton Grange, Northallerton.
Mr. Wm. Cochrane, Grainger Street West, Newcastle-upon-Tyne.
Mr. J. B. Simpson, Hedgefield House, Blaydon-upon-Tyne.

Past-Presidents.

Mr. W. Armstrong, Pelaw House, Chester-le-Street. — *Retiring Vice-President.*

The Chairman said he regretted very much that Mr. Steavenson was not present to receive his congratulations on having been elected to the Presidential chair. Mr. Steavenson ought to have been elected many years ago, but he had always declined to stand owing to his deafness. However, the Council had prevailed upon him to allow himself to be nominated on their promising to assist him in every way to overcome his defect. Mr. Steavenson, his successor, was well known to all; he had contributed some very valuable papers on mining and ventilation, and he had done very much to enhance the prosperity of the Institute.

Mr. G. B. Forster moved a vote of thanks to Mr. J. B. Simpson on his retiring from the office of President, but regretted that they should lose him. They would all agree that Mr. Simpson had conducted the affairs of the Institute in an able and energetic manner during his term of office. The Institute had taken a higher place now and during the past few years than it had ever occupied before. This was shown by its connexion with the Federated Institution of Mining Engineers in comparison with the other Institutes. He believed a very large number of members were joining this Institute so that they might become members of the Federated Institution of Mining Engineers, and in choosing this Institute they showed that they considered it to be the best. They were all sorry to lose him, although he quite endorsed what Mr. Simpson had

said about his successor (Mr. Steavenson). He begged to propose that their most cordial thanks be given to the retiring President (Mr. Simpson), and also to the retiring Vice-Presidents and Council.

Mr. J. G. WEEKS desired to echo Mr. Forster's remarks. While they were sorry to lose Mr. Simpson from the Presidential chair, yet he had some pleasure in thinking they were releasing him from what had been a strain —though a pleasurable one—upon his time and attention. He begged to second the motion.

The vote of thanks was heartily adopted.

Mr. J. B. SIMPSON replying, thanked Mr. G. B. Forster and Mr. J. G. Weeks for the flattering manner in which they had proposed the vote of thanks and also the members for its cordial adoption. He must further thank the Council and members for the kind manner in which they had supported him during his term of office. He could only say it had been a great pleasure to him to be President of the Institute, but he was afraid he had not done his duty as he might, though what he had failed in doing was from want of knowing how to do better. He could only say his love for the Institute would not abate when he left the Presidential chair, and he hoped he would be able to assist in its maintaining its position in the future.

Mr. W. C. BLACKETT moved, and Mr. J. B. SIMPSON seconded, a resolution thanking the scrutineers for their services, which was adopted.

The SECRETARY read the annual reports of the Council and Finance Committee as follows :—

ANNUAL REPORT OF THE COUNCIL.

The following table exhibits the progress of the number of the membership during the two past years :—

	August 1st.		
	1891.	1892.	1893.
Honorary Members	24	24	25
Members	535	545	565
Associate Members	29	35	47
Associates	28	34	47
Students	36	35	34
Subscribing Collieries	18	18	22
Totals	670	691	740

Ninety-eight members of all classes have joined the Institute during the year, and after allowing for the losses of members through death, etc., there remains a net increase of 49 members. While congratulating the members upon the increasing prosperity of the Institute, the Council would especially ask them at all times to promote its claims and advantages, and to extend its membership.

The connexion with the Federated Institution of Mining Engineers has now existed for four years, and the character of the papers communicated at its well-attended meetings proves that its success is now assured. The Mining Institute of Scotland has now become federated, and it is hoped that the remaining non-federated mining and metallurgical institutes may join at early dates.

The library has been maintained in an efficient condition during the year. The additions by donations, exchange, and purchase have been :—

Bound volumes	260
Pamphlets, reports, etc.	234
A total of	494 titles.

And there are now at least 6,401 volumes and 1,022 unbound pamphlets in the library.

Mr. Thomas Dacres has presented two MS. volumes of sections of Borings and Sinkings, and the Council have pleasure in recording the thanks of the Institute to him for these valuable records.

The sets of some of the Transactions of Societies, etc., in the library are incomplete owing to loss of volumes or parts, and it is urgently desired that members will return any volumes, etc., in their possession to the librarian.*

Members could also render useful service to the profession, by presentations of books, reports, plans, etc., to the Institute, where they would be preserved in the library, and be available for reference.

The report of the committee on mechanical ventilators is being completed, and the Council trust that it will be shortly issued to the members.

During the course of the year, it was deemed desirable to dissolve the then existing explosives committee, and a new committee was appointed to continue the investigations and to report upon the so-called "flameless explosives." The committee anticipate that they will be able to issue a preliminary report of their proceedings during the course of the year.

* See list in Annual Report of the Council for 1892.

The fifth volume of the Borings and Sinkings is now approaching completion, and the Council trust that the concluding volumes will be issued shortly.

The Institute has again been placed on the list of corresponding societies of the British Association, and Prof. J. H. Merivale has been appointed to act as delegate at the ensuing meeting at Nottingham.

The papers communicated to the Institute have been :—

"Use of Cement in Shaft-sinking." By Mr. Bennett H. Brough.

"Observations on Fans of Different Types working on the same Upcast Shaft." By the Rev. G. M. Capell.

"Manometric Efficiency of Fans." By the Rev. G. M. Capell.

"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment." By Mr. A. G. Charleton.

"Steam Boilers with Forced Blast : the Perret System for Burning Dust and Rejected Fuels ; with notes on Testing Boilers." By Mr. Bryan Donkin, Jun.

"The Electric Transmission of Power and its Analogies in Hydraulics." By Prof. Wm. Garnett.

"Notes on the Occurrence of Manganese Ore near the Arenigs, Merionethshire." By Mr. Edward Halse.

"Notes on the Occurrence of Manganese Ore near Mulegé, Baja California, Mexico." By Mr. Edward Halse.

"The Gold-bearing Veins of the Organos District, Tolima, U.S. Colombia." By Mr. Edward Halse.

"The Education of Mining Engineers." By Prof. J. H. Merivale.

"The Ordnance Maps." By Prof. J. H. Merivale.

"Joseph Moore's Hydraulic Pumping Arrangement." By Mr. R. T. Moore.

"The Geology and Coal-deposits of Natal." By Mr. R. A. S. Redmayne.

"Gold-mining in Brazil." By Mr. E. M. Touzeau.

"Queensland Coal-mining ; and the Method adopted to Overcome an Underground Fire." By Mr. E. S. Wight.

"The Coad Electric Miner's Lamp." By Mr. Henry White.

The unfortunate fire in the buildings of the Literary and Philosophical Society resulted in no damage to the premises of this Institute, which so closely adjoins.

An agreement has been sealed with the North Eastern Railway Company for the use of an entrance from Orchard Street, and the occupation of a broad passage adjoining the basement of the Wood Memorial Hall.

FINANCE REPORT.

The ordinary income for the year 1892-93 amounted to £1,811 14s. 3d., being an increase over the ordinary income of the previous year of £83 18s. 5d.

The total receipts from subscriptions and arrears (excluding sums paid in advance) amounted to £1,460 11s. 0d., an increase of £74 19s. 0d. over the preceding year.

The total amount of subscriptions now in arrear (including those for the year 1892-93) is £220 10s. 0d., which, it is expected, will be fully paid. The sum of £264 12s. 0d., extending over nine years, has been struck off during the year as irrecoverable arrears.

The total expenditure amounted to £1,732 15s. 6d., a decrease of £165 2s. 9d. on the previous year, but that year included exceptional payments on account of cleaning the Wood Memorial Hall, and the expenditure of the Explosives Committee. An increase of £210 11s. 0d. in the subscriptions to the Federated Institution is caused by the increased number of members, and the special call made to meet the deficiency on the first three years' working of that Institution.

The CHAIRMAN, in proposing the adoption of the Annual Report, Finance Report, and Accounts, referred with satisfaction to the gradually increasing number of members. They had now 740 members, and he sincerely hoped that in the course of time they would rise to the number of 971 members, which they had in 1877. They would all be aware that, after that period, the North-East Coast Institution of Engineers and Shipbuilders was commenced, and took away a considerable number of the members of this Institute, but he hoped that as time went on they might even reach the position of having 1,000 members. Another point he might mention was the satisfactory progress made by the Federated Institution of Mining Engineers; since the last General Meeting the Mining Institute of Scotland had become Federated, and it was hoped that the remaining institutes in the country would follow that example, and that there would then be one vast society—an Imperial Institution of Mining Engineers. It would be noticed that the Flameless Explosives Committee were still proceeding with their investigations; so far the committee had done very good work, but they still required more money to

continue their experiments, and the Council had to-day proposed that they should have a grant of £100 for this purpose. It would be noticed that a large amount of arrears had been written off as irrecoverable, but it must not be concluded that this was for the past year only, it represented the accumulations of nine years; the Council had, however, made such arrangements as would, it was hoped, obviate this in future, and the question of arrears would be more closely followed up. The increase of £210 11s. 0d. in the payments to the Federated Institution of Mining Engineers was caused by increased number of members, and by a special call to meet the balance of the first three years' expenditure. It was estimated that 15s. per member would meet all requirements, and it was found that the expenditure amounted to about 16s. per member, but it was hoped that after the end of the current year a call of 15s. per head would be sufficient. The Council had thought it advisable to appoint a committee to look into the question of their expenditure, with a view to seeing if it could be reduced, as it was thought that the expenditure bore rather a large proportion to their income. He begged to move the adoption of the reports and accounts.

Mr. Thomas Douglas expressed his pleasure in seconding the adoption of the reports. It occurred to him that at a meeting of this kind it was well to mention a source of income which they might tap to a greater extent—he referred to the amounts subscribed by colliery proprietors to the Institute. It appeared that such subscriptions did not amount to £100 a year, which was a very small sum when they remembered the large benefit that must accrue to the colliery proprietors from the information laid before them by the various gentlemen well calculated to give it. The Explosives Committee, for instance, were making experiments which would prove of immense value to colliery proprietors, and he thought they might fairly impress upon them that they should support an institution such as this which afforded information of such value to themselves. He mentioned this question so that the agents who were present might induce their respective proprietors to support the Institute.

The reports were adopted.

DR. THE TREASURER IN ACCOUNT WITH THE NORTH OF ENGLAND FOR THE YEAR ENDING

July 23, 1892.	£	s.	d.	£	s.	d.
To Balance at Bankers	448	14	2			
„ „ in hand	36	18	4			
				485	12	6
„ Dividend of 3¾ per cent. on 134 Shares of £20 each in the Institute and Coal Trade Chambers Co., Ltd., for the half-year ending December, 1892	100	10	0			
„ „ 3¾ for half-year ending June, 1893 ...	100	10	0			
„ Interest on Investments with the River Tyne Commissioners	53	12	6			
				254	12	6
TO SUBSCRIPTIONS FOR 1892-93 AS FOLLOWS:—						
131 Members ... ... @ £2 2s.	905	2	0			
26 Associate Members ... ... @ £2 2s.	54	12	0			
28 Associates ... ... @ £1 1s.	29	8	0			
21 Students ... ... @ £1 1s.	22	1	0			
56 New Members ... ... @ £2 2s.	117	12	0			
14 „ Associate Members ... ... @ £2 2s.	29	8	0			
16 „ Associates ... ... @ £1 1s.	16	16	0			
5 „ Students ... ... @ £1 1s.	5	5	0			
	1,180	4	0			
TO SUBSCRIBING COLLIERIES, VIZ.:—						
Ashington Coal Company £2 2 0						
Birtley Iron Company 6 6 0						
Bridgewater Trustees 6 6 0						
Marquis of Bute 10 10 0						
Earl of Durham 10 10 0						
Haswell Coal Company 4 4 0						
Hetton Coal Company 10 10 0						
Hutton Henry Coal Company 2 2 0						
Marquess of Londonderry 10 10 0						
North Brancepeth Coal Company ... 2 2 0						
Owners of North Hetton Colliery ... 6 6 0						
Ryhope Coal Company 4 4 0						
Seghill Coal Company 2 2 0						
South Hetton Coal Company 4 4 0						
Stella Coal Company 2 2 0						
Owners of Throckley Colliery 2 2 0						
Victoria Garesfield 2 2 0						
Wearmouth Colliery 4 4 0						
	92	8	0			
„ 1 NEW SUBSCRIBING COLLIERY—						
Butterknowle Coal Company 2 2 0						
	2	2	0			
	1,274	14	0			
Less—Subscriptions for current year paid in advance last year	23	2	0			
	1,251	12	0			
To Arrears	208	19	0			
	1,460	11	0			
„ Subscriptions paid in advance during current year ...	53	11	0			
				1,514	2	0
„ Sale of Publications				42	19	9
				£2.297	6	9

INSTITUTE OF MINING AND MECHANICAL ENGINEERS. CR.
JULY 31, 1893.

July 22, 1893.	£	s.	d.	£	s.	d.
By Stationery Account	157	3	9			
„ Books for Library	74	1	3			
„ Prizes for Papers	36	15	0			
„ Incidental Expenses	53	19	4			
„ Sundry Accounts	14	6	6			
„ Travelling Expenses	0	8	6			
„ Salaries	150	0	0			
„ Clerks' Wages	143	12	8			
„ Reporter's Salary	14	14	0			
„ Rent	95	5	8			
„ Rates and Taxes	17	19	10			
„ Insurance	8	15	9			
„ Furnishing, Repairs, etc.	26	4	9			
„ Coals, Gas, and Water	22	6	0			
„ Postages	45	5	2			
„ General Index	15	19	6			
				876	17	8
„ Fan Committee	2	19	0			
„ Explosives Committee	85	1	1			
„ British Association Meeting—Delegate's Expenses ...	6	9	0			
				94	9	1
By Federated Institution of Mining Engineers—Subscriptions	791	15	10			
Less—Amounts repayable by Authors for Excerpts ...	30	7	1			
				761	8	9
				1,732	15	6
By Balance at Bank	468	3	8			
„ „ in hand	84	12	3			
„ Outstanding Amounts by Authors for Excerpts ...	11	15	4			
				564	11	3
				£2,297	6	9

I have examined the above account with the Books and Vouchers relating thereto, and certify that, in my opinion, it is correct.

JOHN G. BENSON,
CHARTERED ACCOUNTANT.

Newcastle-upon-Tyne,
August 4th, 1893.

DR. THE TREASURER IN ACCOUNT

			£	s.	d.	£	s.	d.
To 544 Members,								
24 of whom are Life Members.								
520	520	@ £2 2s.				1,092	0	0
To 35 Associate Members,								
5 of whom are Life Members.								
30	30	@ £2 2s.				63	0	0
To 34 Associates		@ £1 1s.				35	14	0
To 37 Students,								
1 transferred to list of Members.								
36	36	@ £1 1s.				37	16	0
To 18 Subscribing Collieries		... ...				92	8	0
To 56 New Members		@ £2 2s.				117	12	0
To 14 New Associate Members		@ £2 2s.				29	8	0
To 16 New Associates		@ £1 1s.				16	16	0
To 5 New Students		@ £1 1s.				5	5	0
To New Subscribing Collieries ...		... ...				6	6	0
						1,496	5	0
To Arrears, as per Balance Sheet 1891-92		... ...	470	8	0			
Deduct 6 Students transferred to Members		@ £1 1s. ...	6	6	0			
			464	2	0			
Less—Struck off as irrecoverable. Arrears		£206 17 0						
„ „ „ Current year		57 15 0						
			264	12	0			
						199	10	0
						1,695	15	0
To Subscriptions for previous year paid by New Members			...		...	8	8	0
„ Subscriptions Paid in Advance			...		...	53	11	0
						£1,757	14	0

WITH SUBSCRIPTIONS, 1892-93. CR.

		PAID. £ s. d.	UNPAID. £ s. d.
By 431 Members, paid	@ £2 2s.	905 2 0	.....
By 87 „ unpaid	@ £2 2s.	.....	182 14 0
By 2 „ dead	@ £2 2s.	.....	4 4 0
520			
By 26 Associate Members, paid	@ £2 2s.	54 12 0	.....
By 4 „ „ unpaid	@ £2 2s.	.....	8 8 0
30			
By 28 Associates, paid	@ £1 1s.	29 8 0	.....
By 6 „ unpaid	@ £1 1s.	.....	6 6 0
34			
By 21 Students, paid	@ £1 1s.	22 1 0	.....
By 15 „ unpaid	@ £1 1s.	.....	15 15 0
36			
By Subscribing Collieries		92 8 0	.....
By 56 New Members, paid	@ £2 2s.	117 12 0	.....
By 14 New Associate Members, paid	@ £2 2s.	29 8 0	.....
By 16 New Associates, paid	@ £1 1s.	16 16 0	.....
By 5 New Students, paid	@ £1 1s.	5 5 0	.....
By 1 New Subscribing Colliery, paid		2 2 0	.....
By 1 „ „ unpaid		.....	4 4 0
		1,274 14 0	221 11 0
Less struck off as irrecoverable		.....	57 15 0
			163 16 0
By Arrears, paid		208 19 0	.....
„ „ unpaid		.....	56 14 0
By Subscriptions paid in advance		53 11 0	.....
		1,537 4 0	220 10 0
			1,537 4 0
			£1,757 14 0

LIABILITIES.	£	s.	d.	£	s.	d.
Subscriptions paid in Advance during the year	53	11	0			
„ carried over from previous year	4	4	0			
				57	15	0
Capital				9,394	8	9
				£9,452	3	9

I have examined the above account with the books and vouchers relating thereto, and certify that, in my opinion, it is correct. The Share Certificates and Bonds have been produced to me.

JOHN G. BENSON,
CHARTERED ACCOUNTANT.

Newcastle-upon-Tyne,
August 4th, 1893.

ASSETS.	£	s.	d.	£	s.	d.
Balance of Account at Bankers	468	3	8			
„ in Treasurer's hands	84	12	3			
Outstanding amounts for Authors' Excerpts ...	11	15	4			
				564	11	3
134 Shares of £20 each in the Institute and Coal Trade Chambers Co., Ltd.	.....			2,680	0	0
Invested with River Tyne Commissioners ...	.....			1,500	0	0
Arrears of Subscriptions	.....			220	10	0
Value of 548 Bound Volumes of Transactions, @ 8s. 6d.	232	18	0			
„ 4,460 Sewn do., @ 6s.	1,338	0	0			
„ 2,421 Unbound Parts, @ 1s.	121	1	0			
„ 77 Copies of Mr. T. F. Brown's Map, @ 2s. 6d.	9	12	6			
„ 375 Copies of Index, Vols. 1-25, @ 1s.	18	15	0			
„ 849 Copies of Catalogue of Fossils, @ 2s. 6d.	106	2	6			
„ 755 Copies of Fossil Illustrations, @ 5s.	188	15	0			
„ 1,500 Copies of Borings and Sinkings (Vol. I. in Sheets) @ 1s.	75	0	0			
„ 275 Do., Vol. I., @ 2s. 6d.... ...	34	7	6			
„ 304 Do., Vol. II., @ 2s. 6d.... ...	38	0	0			
„ 317 Do., Vol. III., @ 2s. 6d.... ...	39	12	6			
„ 429 Do., Vol. IV., @ 2s. 6d.... ...	53	12	6			
„ 263 Copies of Library Catalogue, @ 1s.	13	13	0			
„ 313 Copies of French Commission Report on Explosives in parts, @ 1s. ...	15	13	0			
„ 10 Do., Bound, @ 4s.	2	0	0			
				2,287	2	6
„ Office Furniture and Fittings ...	450	0	0			
„ Books and Maps in Library	1,750	0	0			
				2,200	0	0
				£9,452	3	9

REPRESENTATIVES ON THE COUNCIL OF THE FEDERATED INSTITUTION OF MINING ENGINEERS.

The SECRETARY stated that the Council had nominated the following gentlemen for election:—

Mr. WM. ARMSTRONG Jun.
Mr. T. W. BENSON.
Mr. THOS. J. BEWICK.
Mr. WM. COCHRANE.
Mr. JNO. DAGLISH.
Mr. THOS. DOUGLAS.
Mr. T. E. FORSTER.
Mr. J. L. HEDLEY.
Mr. T. HEPPELL.
Mr. HENRY LAWRENCE.
Mr. W. LISHMAN (B.H.).
Mr. GEO. MAY.
Prof. J. H. MERIVALE.
Mr. M. W. PARRINGTON.
Mr. J. B. SIMPSON.
Mr. A. L. STEAVENSON.
Mr. T. H. M. STRATTON.
Mr. J. G. WEEKS.
Mr. W. O. WOOD.

The CHAIRMAN moved and Mr. COCHRANE seconded the motion, which was unanimously agreed to, that these gentlemen be elected as the representatives of this Institute for the ensuing year.

Mr. T. H. M. STRATTON proposed that a hearty vote of thanks be accorded to those gentlemen who had represented the Institute on the Council of the Federated Institution during the past year. Any member of this Council who proposed to devote the proper amount of time and attention to it would do so at considerable sacrifice. The meetings were held at such times and places as rendered it very difficult for the representatives from this district to attend, unless they were masters of their own time, and no doubt those members who had been struck off the list as ineligible for re-election owing to non-attendance would have been present if it were possible; and to those who had attended the thanks of the Institute were especially due. The Council had done him the honour to include his name in the list of representatives for the ensuing year, and he made these remarks fearing lest he might be deprived from attending.

Mr. JOHN SIMPSON seconded the vote of thanks, which was unanimously adopted.

Mr. WILLIAM COCHRANE said he was certain those members of the Council who had attended the meetings would be glad to feel that what they had done met with the approbation of the members. Mr. Stratton's remarks raised a point of some importance; the fact of the meetings being frequently held in the Midland Counties rendered it difficult for the members from this district to attend, but they must make an effort to do so, otherwise there would be a much more active interest and influence exerted by the representatives of the institutes which did attend, and this

Institute would be slipping away from the position and influence it had hitherto exercised as the leading Mining Institute, and as the originator of the Federated Institution of Mining Engineers.

The following gentlemen were elected, having been previously nominated :—

HONORARY MEMBER—

Professor JOHN HERMAN MERIVALE, Professor of Mining, Durham College of Science, Newcastle-upon-Tyne.

MEMBERS—

Mr. ARCHIBALD THOMAS BROWN, Mining Engineer, 372, Flinders Lane, Melbourne.

Mr. WESTGARTH FORSTER BROWN, Mining Engineer, Alston House, Parade, Cardiff.

Mr. RICHARD ECK, Mining and Mechanical Engineer, Phœnix Hotel, Beaconsfield, South Africa.

Mr. JOSEPH GOULDIE, Mining and Civil Engineer, The North-Eastern Bulfontein, Kimberley, South Africa.

Mr. JOSEPH HARGREAVES, Colliery Manager, Gwaun Cae Gurwen Collieries, Brynamman, R.S.O.

Mr. JOHN HOLT, Jun., Civil and Mining Engineer, The Hollies, Heywood, Lancashire.

Mr. EDWARD HOPPER, Engineer, c/o Messrs. Lewis & Marks, Coal Mines, Vereeneging, Transvaal.

Mr. JOHN WILSON RICHMOND LEE, Mining Engineer, Potes, Provincia de Santander, Spain.

Mr. J. B. ROBINSON, Mining Engineer, Hedley Hill Colliery, Waterhouses.

Mr. WALTER ROWLEY, Mining Engineer, 20, Park Row, Leeds.

Mr. THOMAS BIRCH FREEMAN SAM, Mine Manager, Adjah Bippo, West Coast, Africa.

Mr. JOSEPH SCOTT, Mining Surveyor, Newcastle Street, Stockton, near Newcastle, New South Wales.

ASSOCIATE MEMBER—

Mr. GEORGE THOMAS DUNCAN, Engineer and Agent, 110, Dilston Road, Newcastle-upon-Tyne.

ASSOCIATES—

Mr. EVAN COCKBURN, Back Overman, Page Bank Colliery, *via* Willington, Co. Durham.

Mr. WM. HENDERSON, Engineer, Wheatley Hill Colliery, *via* Trimdon Grange.

Mr. MOSES HOBSON, Under Manager, Shildon, *via* Darlington.

The following gentlemen were nominated for election :—

MEMBERS—

Mr. JOHN WATSON COOK, Colliery Manager, Greenfield House, Crook.

Mr. LEWIS EVANS, Mining Engineer, Coetzeestroom Estate and Gold-Mining Co., Ltd., Kaapsche Hoop, North Dekaap, Transvaal.

Mr. FRANK SIMPSON FERENS, Mechanical and Electrical Engineer, 13, Railway Arches, Westgate Road, Newcastle-upon-Tyne.

Mr. HENRY J. GIFFORD, Superintendent of the Ouro Preto Mines of Brazil, Ltd., Mines de Passagem, Ouro Preto, Brazil.

Mr. FREDERICK HALL, Mines Manager, Harrington Road, Workington.

Mr. JOHN HEDLEY, Mining Engineer, Coatham, Redcar.

Mr. JOHN CARY BAKER HENDY, Mining Engineer, Etherley, by Darlington.

Mr. FRANK T. HOWES, Assistant Colliery Manager, Singareni Collieries, Hyderabad, Deccan, India.

Mr. WILLIAM HUMBLE, Government Inspector of Collieries, Coal-fields Office, Newcastle, New South Wales.

Mr. WILLIAM HAMILTON MERRITT, Mining Engineer and Metallurgist, 485, Huron Street, Toronto, Ontario, Canada.

Captain G. NICHOLLS, H.M. Commissioner of Mines for Natal and Zululand, Pietermaritzburg, Natal.

Mr. RALPH NIXON, Lecturer on Mining to Northumberland County Council, 47, Holly Avenue, Newcastle-upon-Tyne.

Mr. JOHN SHOTTON, Assistant Engineer, Locomotive Department, Ottoman Railway Co., Smyrna, Asia Minor.

Mr. JAMES JOSEPH TONKIN, Manager of Silver-lead Smelting Works, Linares, Provincia de Jaen, Spain.

ASSOCIATE MEMBERS—

Mr. HERMAN ALEXANDER KROHN, Secretary to the Roburite Explosives Co., Ltd., 103, Cannon Street, London, E.C.

Mr. GEORGE FREDERICK MANSELL, Public Accountant and Fellow of the Institute of Secretaries, St. Mary's Chambers, Church Street, Colchester, Essex.

ASSOCIATES—

Mr. J. T. BOLAM, Overman, The View, Beamish, Chester-le-Street.

Mr. JOHN FAIRS, Under Manager, Escomb Bridge, Bishop Auckland.

STUDENT—

Mr. ALGERNON NOBLE, Mining Apprentice, Broomhill Colliery, Acklington.

Mr. C. C. LEACH read the following paper on a "Corliss-engined Fan at Seghill Colliery":—

CORLISS-ENGINED FAN AT SEGHILL COLLIERY.

By C. C. LEACH.

The ventilation at Seghill colliery was, until recently, produced by two furnaces, one placed in the yard seam and the other in the low main seam, burning 1,310 tons of coal per annum.

It was decided to erect a fan capable of doubling the volume of air then circulating in the blake and yard seams.

The calculated power of the engine to do this was 120 indicated horse-power, and a fan 35 feet in diameter running 61 revolutions per minute.

The following table contains the results of experiments, as compared with the preliminary calculations:—

TABLE I.

Name of Seam.	Depth from the Surface.	Volume of Air produced by Furnace per Minute.	Calculation of the Volume of Air produced by Fan running at 61 Revolutions per Minute.	Volume of Air per minute produced by Fan when running at 60 Revolutions per Minute in the return Air-ways.
	Feet.	Cubic Feet.	Cubic Feet.	Cubic Feet.
Blake ...	210	24,460	48,920	66,721
Yard ...	275	23,222	46,440	45,567
Low main	421	15,028	15,028	9,996
Total volume of air ...		62,710	110,388	122,284
Total horse-power in air		10·87	58·94	52·65
Indicated horse-power of engine			120·00	96·11

The above table shows that less speed and power is required to produce the desired volume of air by the fan than was calculated, but this result was expected, and is due to the openings into the shafts and airways being improved, and to the fan and engine yielding 4·78 per cent. higher useful effect than was anticipated, besides any fractional part was set against the engine.

Most fan engines are far too large for the work they are doing, and therefore have very low initial pressures on the piston, the steam being on for nearly the full stroke, but throttled down by the stop-valve to sometimes even 8 or 10 lbs. per square inch, and owing to the absence of governors the speed varies as the boiler pressures rise and fall. These large engines have a great amount of extra weight to keep moving, they require larger steam pipes, and have large radiating surfaces.

To overcome these defects and to obtain the most economical engine possible with one cylinder, the Corliss engine and a steam pressure of 100 lbs. per square inch was selected; because in the writer's opinion it is the highest pressure that can be used with advantage in an engine with one cylinder, and affords great range of economic power in this type of engine with its variable automatic cut-off.

The fan has been at work fifteen months; it is open-running, 35 feet 1 inch in diameter, the inlet is $12\frac{1}{2}$ feet in diameter, and the inside width at the rim is $15\frac{1}{2}$ inches. It is made of extra thick steel plates, the front plates being $\frac{3}{16}$ inch thick, the back plates $\frac{1}{4}$ inch, the back plates opposite the inlet are $\frac{7}{16}$ inch, and the vanes $\frac{3}{16}$ inch, and it weighs about 19 tons. The fan shaft is mild steel 8 feet 8 inches long between the bearings, $11\frac{1}{2}$ inches in diameter, and where the fan bosses are keyed on it is 12 inches in diameter. The drift bearing (which carries two-thirds of the weight) is 10 inches in diameter and 20 inches long; the bearing in the engine-house is $8\frac{1}{2}$ inches in diameter and 14 inches long. The fan and shaft were made by Messrs. Thornewill & Warham.

The engine was built by Messrs. Hick, Hargreaves, & Co., and has one horizontal high-pressure cylinder, 16 inches in diameter and 3 feet stroke, with separate end-valve chambers, separate internal cylinder-barrel of special cast-iron, sight-feed lubricator, and steam-jacketed. Both the cylinder and its covers are felted and lagged. The steam and exhaust valves are Corliss valves, worked by the Inglis and Spencer gear; there are double eccentrics and two wrist-plates for working the steam and exhaust valves independently. The wear on the trip-gear is almost inappreciable.

The engine-frame is of box-girder pattern, carrying the crank shaft with phosphor-bronze steps, and having horizontal and vertical adjustments.

The governor controls the speed automatically by varying the cut-off to the speed it is weighted for, from 0 to $\frac{3}{4}$ stroke; the weights can be added or taken off while the engine is running. By adjusting the lever-rods the

governor will allow the steam on for the whole of the stroke continuously, or just now and again as the work done requires.

The valves, each working independently, are very easily set, and almost frictionless, being handed to start the engine with a very short lever. They give the shortest possible ports to and from the cylinder, the waste spaces calculated from the drawings (including $\frac{1}{2}$ inch clearance) is only 5·175 per cent. of the contents of the cylinder.

The piston rod is 3 inches in diameter and works through Cruickshank metallic packing, which is efficient—there is not a scratch on the rod.

The crank pin is $4\frac{1}{2}$ inches in diameter and 5 inches long.

The main bearing in the drift has three weighted piston lubricators and is supplied with solid grease. The engine-house bearing of the fan shaft has circulating lubrication. The crank pin has an oil-syphon and centrifugal lubrication.

Calculated from the result of 225 days' working, the stores used by the engine per year are as under :—

	£	s.	d.
42 gallons of cylinder oil, at 2s.	4	4	0
18 gallons of engine oil, at 1s.	0	18	0
$14\frac{4}{5}$ gallons of castor oil, 130 lbs., at $3\frac{1}{2}$d.	1	17	11
133 lbs. of waste, at 2d.	1	2	2
16 sheets of emery cloth, at 1d.	0	1	4
Yearly cost of stores for engine	8	3	5
114 lbs. Delettrez grease for drift-bearing,* at 1s. 4d....	7	12	0
Total	£15	15	5

The engine gives neither knock nor heat on the bearings, although there is about 9 tons of push each time the steam is admitted at every speed ; with attention, it gives the engineman no trouble.

The engine and foundations are adapted for a condenser being added, if more power be required, but at present, at 40 revolutions, the engine is too big for its work.

The steam pipes are 103 feet long and 7 inches in diameter, and 14 feet of 5 inches in diameter pipes with ten right-angled bends. Other engines are supplied by the 7 inches pipes. The exhaust pipes are 9 feet long and 5 inches in diameter, with three right-angled bends. All the steam pipes are covered.

The only true test of the efficiency of an engine is the weight of steam or water required to drive it, for then the type and size of boiler and kind

* Oil did not keep the drift-bearing cool.

of coal burnt does not interfere with the result ; therefore the weight of steam or water used per indicated horse-power per hour is given in Table IV.

The indicator spring was tested with weights, the indicator being screwed upside down on to the steam pipe, and heated during the test with live steam.

The indicator cards were worked out by a planimeter.

The revolutions of the engine were got by a Harding counter and divided by the time to get the revolutions per minute.

A correction of 30 was added to the anemometer revolutions per minute.

The fan drift has an area of 182 square feet, and is divided by wires into 91 squares ; each measurement of the air took fifteen minutes..

There is one downcast pit (11 feet in diameter), with 4 cages ; one downcast pit (8 feet in diameter), with 2 cages and steam pipes ; and one upcast shaft (11 feet in diameter) clear of all obstruction.

A König water-gauge was used when running at 20 revolutions per minute ; and the ordinary water-gauge for the higher speeds. The end of the water-gauge pipe in the drift was 10 feet 4 inches from the centre of the fan shaft, with the end turned towards the fan.

In testing the engine, the water used was pumped from a tank into the boiler.

The specimen indicator cards (Figs. 1, 2, 3, and 4, Plate I.) from the engine, when running at 20, 40, and 60 revolutions per minute, were taken with the mine under ordinary conditions, and at 60 revolutions with the separation-doors open, the air being measured in the drift. Another card (Fig. 5) was taken at 67 revolutions of the fan, the separation-doors being open and full steam turned on the engine for the full stroke, indicating 209·2 horse-power.

The indicator cards were taken by passing the pencil over the same cards three times during each air measurement and about every ten minutes during the steam-consumption tests, the card being changed every hour, and the averages taken.

On April 25th, 1892, the front steam-valve was disconnected and the steam turned full on the engine again ; the governor kept the engine running for $5\frac{1}{2}$ hours at the same speed (35 revolutions), although the steam was only admitted to the back end of the cylinder, the indicated horse-power being about the same as the previous day, with both valves at work (Fig. 6, Plate I.).

TABLE II.

Date of Experiment	April 29th, 1893.	May 11th, 1893.				May 15th, 1893.				Average of Experiments, 2 to 9.
No. of Experiment	1.	2.	3.	4.	5.	6.	7.	8.	9.	
Condition of mine	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.
Revolutions of fan per minute	19·87	39·97	39·87	39·63	39·90	39·52	39·89	39·72	39·95	39·806
Revolutions of anemometer per minute	242	448	443	446·5	454	460	458	455	461	—
Velocity of air in feet per minute	272	478	473	476·5	484	490	488	485	491	—
Volume of air in drift in cubic feet per minute	49,686	86,996	86,086	86,723	88,088	89,180	88,816	88,270	89,362	87,940
Water-gauge in drift in inches	0·29	1·13	1·133	1·12	1·103	1·06	1·06	1·06	1·06	1·09
Temperatures in air, wet bulb	—	54¼	53	52	51½	46½	46½	46½	46½	—
„ „ dry bulb	45½	62½	61½	59	58¼	47½	47¼	47½	47¼	—
„ in drift, wet bulb	—	60	59¾	59¼	59¼	60	60½	59½	59½	—
„ „ dry bulb	60½	60	60	59½	59½	60	60½	59½	59¾	—
Boiler pressure, pounds per square inch	60	99	100	100	99	99	100	99	100	—
Mean pressure of indicator-diagrams in pounds per square inch	9·04	20·26	20·565	20·28	20·224	21·09	21·00	21·17	21·21	—
Weight of an inch of water in pounds	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2
Horse-power in air	2·27	15·49	15·36	15·305	15·31	14·89	14·83	14·74	14·92	15·105
Indicated horse-power of engine	6·463	29·08	29·44	28·86	28·97	29·86	30·08	30·19	30·42	29·612
Efficiency, per cent.	35·12	53·26	52·17	53·03	52·84	49·80	49·30	48·82	49·05	51·02

TABLE III.

Date of Experiment	May 17th, 1893.				May 25th, 1893.				Average of Experiments, 10 to 17.	May 14th, 1893.		
No. of Experiment	10.	11.	12.	13.	14.	15.	16.	17.		18.	19.	20.
Condition of mine	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	Ordnry.	*	*	*
Revolutions of fan per minute	60·83	60·82	60·68	61·00	59·96	60·42	60·08	59·72	60·44	60·00	59·88	67
Revolutions of anemometer per minute	746	734	723	734	694	697·5	704·2	694	—	1,330	1,332	—
Velocity of air in ft. per minute	776	764	753	764	724	727·5	734·2	724	—	1,360	1,362	—
Volume of air in drift in cubic feet per minute	141,232	139,048	137,046	139,048	131,768	132,405	133,624	131,768	135,742	247,520	247,884	—
Water-gauge in drift in inches	2·46	2·47	2·46	2·48	2·42	2·44	2·50	2·46	2·46	2·14	2·14	—
Temperatures in air, wet bulb	48	48	48½	48	49½	49¼	48¼	48	—	52	52	—
„ „ dry bulb	48½	47½	48½	48	54½	54½	52	51½	—	55½	55½	—
„ in drift, wet bulb	—	60	59½	—	58½	58½	58¼	58	—	56	56	—
„ „ dry bulb	—	60	59½	—	59	59	58¾	58½	—	57½	57½	—
Boiler pressure in lbs. per sq. in.	99	99	98	100	97	97	99	97	—	106	106	—
Mean pressure of indicator diagrams in pounds per sq. in.	45·34	45·04	44·87	45·02	43·428	43·468	43·880	43·203	—	71·32	72·10	86·95
Weight of an in. of water in lbs.	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2	5·2	—
Horse-power in air	54·746	54·119	53·124	54·338	50·247	50·907	52·64	51·08	52·65	83·47	83·59	—
Indicated horse-power of engine	99·04	98·37	97·76	98·62	93·505	94·31	94·66	92·65	96·114	153·67	155·03	209·2
Efficiency per cent.	55·27	55·01	54·34	55·09	53·73	53·99	55·61	55·13	54·78	54·32	53·92	—

* Separation-doors open.

TABLE IV.

Date of Experiment	May 8th, 1893.	May 9th, 1893.	May 23rd, 1893.	May 29th, 1893.	Average of Four Tests, A, B, C, and D.	May 6th, 1893.	May 10th, 1893.	May 25th, 1893.	Average of Three Tests, E, F, and G.
Experiment	A.	B.	C.	D.		E.	F.	G.	
Duration of test in hours	7½	8¼	8	7⅓	7·77	6	8·633	8	7·544
Revolutions of engine per minute	39·91	39·84	39·77	40·20	39·93	60·13	60·18	60·013	60·108
Boiler pressure in pounds per square inch	98	99	99	99	98	96	98	97	96
Mean pressure of indicator diagrams in pounds per square inch	22·39	22·30	20·736	22·116	—	46·556	46·64	43·584	—
Indicated horse-power of engine	32·15	31·97	29·61	31·92	31·41	100·52	101·00	93·925	98·482
Water put into boiler, in pounds	7,938	10,584	8,883	7,938	—	16,065	23,814	21,168	—
Temperature of feed water, degs. Fahr.	128	129	118	135	128	115	141	172	143
Waste water in pounds	172	155	136	150	—	186	177	224	—
Water used by engine and steam jacket in pounds	7,766	10,429	8,747	7,788	8,682	15,879	23,637	20,944	20,153
Water from steam jacket in pounds	401	501½	467	418	—	470·5	562	476	—
Steam jacket water, per cent.	5·16	4·81	5·33	5·37	5·17	2·97	2·37	2·27	2·54
Steam or water used per indicated horse-power per hour, in pounds	32·19	39·54	36·92	33·27	35·57	26·32	27·1	27·87	27·12

TABLE V.—SUMMARY OF EXPERIMENTS.

Revolutions of Fan per Minute.	Useful Effect.	Volume of Air per Minute in the Fan Drift.	Indicated Horse-power of Engine.	Steam or Water used per Indicated Horse-power per Hour. (Including Steam Jacket.)		Jacket Water.	Condition of Mine.
				Measured into Boiler.	Accounted for by Diagram Card.		
	Per Cent.	Cubic Feet.		Pounds.	Pounds.	Per Cent.	
20	35·12	49,686	6·46	—	—	—	Ordinary.
40	51·02	87,940	29·61	35·57	24·34	5·17	„
60	54·78	135,742	96·11	27·12	21·50	2·54	„
60	54·12	247,702	154·35	—	—	—	Separation doors open.
67	—	—	209·20	—	—	—	

The tests prove great economy of steam, unparalleled regularity of fan revolutions, that there is no wire-drawing, and that the back-pressure is under 1 lb., with great range of power.

It is impossible to have a fan engine with a good margin of power for more air, or less favourable condition of the pit, and have its best economy at usual speeds, but it is probable that this engine running at 80 revolutions (its full speed) with a suitable load would give still better results.

Another advantage of this engine is its small size, only requiring small engine-house and small steam pipes.

In conclusion, the writer acknowledges his indebtedness to Mr. M. Walton Brown for his assistance in calculating the size of the fan, and the power required, and also for his assistance when making the experiments recorded in this paper.

The CHAIRMAN asked if Mr. Leach had tried any experiments without the steam jacket, and whether it was a Waddle fan?

Mr. COCHRANE said that about 1,300 tons of coal burnt per annum was given as the quantity formerly used by the furnaces, and that the ventilation had been doubled by the use of the fan. Could Mr. Leach tell them what was the present consumption of coal, so that a comparison could be made? He would also like to ask what additional advantage Mr. Leach expected to obtain by applying a condenser? Judging from the indicator diagrams the economy would be very considerable. Did the ninety-one squares into which the fan drift was divided each contain the anemometer for 15 minutes, or was that the total time of observation?

Mr. Leach said he had tried no experiments without the steam jacket. The fan was of the Waddle type, and took in air at one side only. Fifteen minutes was the total time occupied in measuring the air, and the anemometer was held in each square for about 10 seconds. The boiler, which also supplies steam to the hauling engine, etc., was isolated during the tests, and the small coal actually burnt was weighed. The Lancashire boiler is 8 feet in diameter and 30 feet long, and is about six times too large for the fan engine alone when running at 40 revolutions, and two and a half times too large when running at 60 revolutions per minute. The following table shows the consumption of fuel :—

Fan Revolutions per Minute.	Volume of Air per Minute in Returns.	Duration of Tests.	Fan Engine.		Furnace.	Coals Saved per Annum.
			Small Coal burnt per Annum.	Coal burnt per Indicated Horse Power per Hour.	Small Coal burnt per Annum (calculated).	
	Cubic Feet.	Hours.	Tons.	Pounds.	Tons.	Tons.
40	80,817	31·41	560	4·80	2,804	2,244
60	122,284	22·63	1,437	3·97	9,712	8,275

Very slight advantage would be obtained by using a condenser when running the engine at 40 revolutions per minute (under 100 tons of coal per annum). The condenser was not added because it was wished to start with the engine as simple as possible, and to hold this power of the condenser in reserve.

Mr. W. C. Blackett observed that Mr. Leach had measured the quantity of water consumed during these experiments; it would be interesting to know how he had succeeded in measuring this accurately. Could he also give them any information as to the steam jacket? Was it of any use whatever?

Mr. Leach said the water was pumped from a measured tank into the boiler. The steam jacket was simply intended to keep the cylinder as hot as possible, and to prevent the steam inside the cylinder from being condensed into water.

The Chairman said it was generally understood that a steam jacket gave an additional economy of about 15 per cent.

Mr. Blackett said he failed to see how the use of a larger cooling surface, which the jacket gave, could result in a greater efficiency than the smaller cooling surface of the cylinder alone.

Mr. Leach said the steam had a temperature of 337 degs. on being admitted to the cylinder at 100 lbs. pressure. When it left the cylinder the temperature was about 212 degs., and the live steam next admitted

had to warm the cylinder again; if they could keep it warm on the outside it saved steam inside, especially in high expansions.

The CHAIRMAN said that engineers were very much divided as to the efficiency of the steam jacket, and during the past few years experiments had been made by members of the Mechanical Institution of Engineers, and the results (which had been published in the Transactions of that Institution), he believed, were favourable to the steam jacket. It had always been stated (whether true or not he did not know) that they gained from 10 to 15 per cent. by the adoption of a steam jacket; the steam lost its heat in getting into the cylinder and the jacket acted as a superheater, giving back the heat lost. He had, for his part, always advocated the use of steam jackets, and had attached them to most of the large engines he had put up.

The CHAIRMAN asked Mr. Leach if he could try some experiments without the steam jacket?

Mr. LEACH—It could be done, but it is a special experiment; and the apparatus has been disconnected.

Mr. M. WALTON BROWN did not know whether any member had ever made tests similar to those recorded by Mr. Leach, in which they had compared the quantity of water or steam used with the indicated horse-power. The only other records he had seen were given in the *Transactions* of the Federated Institution of Mining Engineers.* From the results contained in Mr. Hendy's paper on "Experiments upon a Waddle Fan and a Capell Fan Working on the Same Mine at equal Periphery Speeds, at Teversal Colliery," it might be calculated that with the Waddle fan, 119 lbs., and with the Capell fan, 113 lbs. of water or steam were used per indicated horse-power. Mr. Leach's results were much smaller than these, being practically only 30 lbs. of water per indicated horse-power, showing an economy of about 75 per cent. both in the consumption of coal and the water evaporated.

Mr. T. H. M. STRATTON said the difference pointed out by Mr. Brown between these two sets of observations was so great that one could only remark that evidently Mr. Leach had done well in bringing the subject before them. Many engineers were apt to think that the battle was one of fans, when actually it was not so much the fan itself as the circumstances of the mine, the engine driving the fan, and the pressure of steam applied to that engine. He was afraid to think that they had of late years sat there meeting after meeting and heard it asserted that this or that fan was most efficient. Mr. Leach had started a new hare in stating that the best fan was one which used the least volume of water

* Vol. iv., page 476.

per indicated horse-power, and that it was desirable to ascertain what was the best engine to drive a fan, rather than what was the best fan to exhaust the air out of the mine. Mr. Leach had said to him one day that he could not afford to buy a bad engine. He thought there was a great deal of truth in that remark, and if everybody would look upon steam-using with the view that they could not afford to have a bad engine they would obtain much better results from the fan, and from the winding and hauling engines than they had at present.

Mr. R. F. SPENCE asked what was the temperature of the feed-water?

Mr. LEACH replied that it was 150 degs., and the exhaust steam from the engine was used to heat the feed-water.

Mr. HENRY WHITE asked if the same class of coal was burned in each case?

Mr. LEACH—Yes, small coal.

The CHAIRMAN said as no further remarks were offered, he proposed that a vote of thanks be given to Mr. Leach for his very valuable addition to the literature of fan ventilation. It appeared quite clear that high economy could be attained in fan ventilation by the use of economical engines. It was certain that an engine of high or triple-expansion would give a better result than a single cylinder with steam admitted during full length of the stroke. The paper showed the advantage of using such engines and boilers as economized steam and fuel.

Mr. G. B. FORSTER seconded the motion, which was carried with acclamation.

Mr. LEACH, in acknowledging the vote of thanks, said he would be very glad to show the engine and fan to any of the members.

DISCUSSION UPON MR. R. T. MOORE'S PAPER UPON "JOSEPH MOORE'S HYDRAULIC PUMPING ARRANGEMENT."*

Mr. C. J. MURTON asked the author if it were possible to increase the speed of the engine in the same way as the speed of ordinary pumping engines could be increased in cases of emergency, as it seemed to him that any increase of speed would, owing to the restriction met with in the power pipes, be attended with a great loss of efficiency?

The following paper by Mr. Edward Halse on "The Occurrence of Mercury at Quindiú, Tolima, U.S. Colombia," was taken as read:—

* *Trans. Fed. Inst.*, vol. iv., page 331.

NOTE ON THE OCCURRENCE OF MERCURY AT QUINDIÚ, TOLIMA, U.S. COLOMBIA.

By EDWARD HALSE.

According to Humboldt,* a vein of cinnabar was discovered in 1786 near the pass of Quindiú† by some miners from Sapo, thanks to the patriotic zeal of the botanist Mutis, at whose expense the examination was made.

Recently, Señor Joaquin Campuzano examined the colonial archives and discovered a note of the year 1787, to the effect that six workings had been opened up in the Sierra de Quindiú showing veins of cinnabar.‡

In 1886, diligent search was made for the old mine, with the result that in a few months' time indications of cinnabar were discovered, and subsequently the six levels, referred to in the archives, were laid bare, together with various furnaces and some tools.‡

During a two years' residence in Colombia the writer had an opportunity of examining the mine, and it was found to contain some interesting features.

The mine is situated on the eastern slope of the Central Cordillera (known here as La Sierra del Quindiú) of the Andes, one long day's mule journey (25 miles) west-south-west of Ibagué, and a few hours' ride from Ibagué Viejo. The latter place was situated on a small plateau, but now only a few mounds indicate the site of the old town. In the alluvium filling the ravine or *quebrado* of the river Bermillon (Spanish for vermilion) running at the foot of the plateau, rounded fragments of cinnabar were found mixed with pellets of gold.*

The mine is nearly 10,000 feet (3,000 metres) above sea-level, on the side of a steep and densely timbered mountain. At this altitude it rains for

* *Essai Politique sur le Royaume de la Nouvelle-Espagne*, 1811, vol. 4, pages 117-118. See Spanish edition *Ensayo sobre Nueva-España*, Jalapa, 1870, vol. 2, page 101.

† Often spelt Quindió—but the above spelling is correct, and is that given in Mr. J. Arrowsmith's Map of Colombia, and also by Humboldt.

‡ *Revista de Minas*, Bogotá, 1888, No. 2, page 57.

the greater part of the year. The soil is clay, and the trees are festooned with Spanish moss. To anyone coming up from the plains the climate appears damp and disagreeable; nevertheless it is said to be healthy, with a mean annual temperature of about 12 degs. C.

Figs. 1 and 2, Plate II., show the principal workings in sketch-plan and section from a survey made by the writer with a small instrument. Fig. 3, Plate II., is a geological section after Mr. Ezekiel Williamson, who has examined and reported on these deposits.

The formation in which the cinnabar occurs is sedimentary, consisting of chloritic, talco-chloritic, arenaceous and aluminous schists of Palæozoic age (Silurian, Cambro-Silurian or Laurentian). At the foot of the mountain there is a mass of dioritic porphyry forming the bed of the river Bermillon, while near the summit, according to Mr. Williamson, there is a large dyke of diorite. Recent operations show loose diorite at the San Blas and Riveras workings on either side of the supposed dyke. The schists run, on an average, east 30 degs. south to west 30 degs. north, and dip north 39 degs. to 43 degs.* At the time of the writer's visit (June, 1889) the chloritic schists, which are hard and granular, had been proved by the Socabon Español Antiguo for 56 feet—this distance, with the exception of the last 12 feet, having been driven by the Spaniards. At the breast only were there any indications of ore, these consisting of patches of quartz with some cinnabar.

The Socabon Español No. 2 is only about 20 feet vertically above the last, and has been driven about 126 feet as a cross-cut; 40 feet in-bye on the east side a level 14 feet in length has exposed a vein or seam, $3\frac{1}{2}$ feet in width, of clay, iron pyrites, and cinnabar, the centre thereof consisting of layers of soft chloritic schist charged with cinnabar; 37 feet in-bye hard chloritic schists dipping north 30 degs., and in places charged with cinnabar, give place to softer talco-chloritic schists which are in reality the cinnabar-bearing rocks of the district. About 90 feet in-bye there crosses a vein or seam of iron pyrites and cinnabar $3\frac{1}{2}$ feet wide. The roof of the seam consists of $2\frac{1}{2}$ feet of talcho-chloritic schists charged with cinnabar and dipping north 23 degs. This vein appears to dip somewhat flatter than the beds which roll a good deal. At the breast of the level the beds dip north 30 degs. to 45 degs.

The Socabon San Roque is 20 feet above the last, and driven east 9 degs. north, or at an angle of 39 degs. with the average strike of the bedding-planes; 17 feet in-bye on the southern side is a cross-cut driven 33 feet to the south. At the end are from 4 to 5 feet of beds charged with

* All readings magnetic, 1889.

quartz, calcite, and cinnabar—the latter in spots and small strings. In the breast of the level is a flucan running west-north-west, and dipping north 44 degs., showing some cinnabar in the clay of the joint. On the western side of the level near the breast some rather soft beds are exposed with a seam of red-coloured earth 3 or 4 inches thick. The latter, washed in a batea, gave 1½ per cent. of cinnabar.

In the New Almaden mines of California, it was only by carefully following up such colours of mineral that the deposit was developed.*

The Santa Teresa level is to the west of the San Roque, and has been driven for 32 feet. In the end is a vein of very hard quartz, with iron pyrites, calcite, and pretty spots of cinnabar. Below the vein is coloured stuff containing some cinnabar. The beds dip north 25 degs., at the mouth of the level they run about north-west, and dip north-east 39 degs.

Above this level the so-called "veta rica" is seen cropping out at the surface. It contains 2 feet or so of soft decomposed rock with quartz, iron pyrites, oxide of manganese, and cinnabar.

Some little distance above the house, another outcrop of quartz and cinnabar is seen in beds much coloured and spotted with the latter mineral.

Since the writer examined the property, two new levels have been started, but nothing of importance had been discovered down to January 5th, 1890.

A very flat vein of cinnabar was found to crop out just above the New San Blas level running east 30 degs south to west 30 degs. north, dipping south-west 10 degs., and consisting of clay, with bands of quartz impregnated with cinnabar. In the level the rock, dipping south-west 39 degs., consisted of lumps of albitic diorite in felspathic clay containing much iron pyrites.

At the Riveras working, on the other side of the ridge, the rock was dioritic and of the usual strike and dip.

For some depth the rocks in this mine are more or less decomposed to a soft clay; the thickness of the cap varies much: in places the rock is comparatively hard right up to the surface, while elsewhere there exists a cap of soft rock several feet thick.

This so-called alluvial cap is said to have been proved to exist for a length of 5,000 feet (1,500 metres), and a width of 1,300 feet (400 metres), but its value has probably been exaggerated.

* Prof. J. B. Christy, quoted in *Engineering and Mining Journal*, January 5th, 1889, page 10. See also Mr. Becker's monograph. Mr. Coignet says it was by following veinlets of calcite (*Rapport sur les Mines de New Almaden*, 1866, page 11).

Immediately above the house, where an artificial pond (*estanque*) has been cut, the decomposed rock seems to be 15 feet thick. Below it the bed-rock of soft talcose-chloritic schist is seen to trend considerably more to the north than usual. Upon it rests 3 feet of yellow coloured alluvium which gives a little cinnabar in the batea. Above this is red-coloured clay which yielded no cinnabar. The cap tested elsewhere gave sometimes a little cinnabar, and sometimes none at all.

In one place, where the rock is hard at the surface, there is a seam of clay 3 feet thick, which gives some cinnabar in the batea.

The average percentage of cinnabar in the veins and seams examined by the writer was somewhere between one-half and two and a half.

Notwithstanding the fact that the mine could then only be regarded as a prospect—a bold attempt was made to float it in the city of London as a large and proved deposit. Reports were written comparing the mine to the classical Almaden of Spain, and a section was published therewith showing numerous veins of cinnabar (beautifully coloured) traversing the hill from base to summit, a section which one could only compare to those physiological diagrams which illustrate the arterial circulation of the blood.

It is not altogether improbable that a paying deposit occurs farther in the mountain and nearer the large dyke of diorite which is said to penetrate the centre of the hill. But in order to prove this the old Spanish level should be driven on.

At Almaden the cinnabar occurs principally in nearly vertical bands of quartzite or grit, and is comparatively rare in the beds of encasing schist.* Melaphyre is the eruptive rock adjacent to these deposits. Although in one or two places at Quindiú cinnabar is found associated with a quartz-vein or seam, nothing so far has been disclosed approaching to the siliceous cinnabar-impregnated beds of Almaden, so that until something resembling this is discovered the comparison between the two deposits is (to say the least) premature.

For the sake of comparison a table is appended showing the principal known occurrences of cinnabar, compiled mainly from chapter II. of Mr. G. F. Becker's *Geology of the Quicksilver Deposits of the Pacific Slope*.†

The Quindiú deposit may be regarded as impregnations probably intimately connected with the intrusive dyke of diorite. Heated waters bearing double sulphides of iron and mercury in solution have probably made their

* Mr. H. Kuss, *Mémoire sur les Mines et Usines d'Almaden*, *Ann. des Mines*, 1878, page 29.

† U.S. Geological Survey, monograph XIII.

way up and along this disturbance, and, then passing down the bedding-planes of the softer schists, have impregnated certain beds, occasionally filling joints and fissures therein. The reason why they have scarcely penetrated the hard chloritic beds is probably on account of the want of porosity of the latter.

Mr. Becker says: "The mineral associations in which cinnabar is found seems to show conclusively that it has been deposited from solution." The deposits at Quindiú appear to be no exception to this rule. That author further says: "A very large part of the known deposits of cinnabar are extremely similar in character, a fact which seems indicative of a similar origin. It is certain that some of the deposits are due to precipitation from hot volcanic springs, and it may fairly be inferred that many of them were formed in this manner."

COMPARATIVE TABLE OF CINNABAR DEPOSITS.

Locality.	Rock.	Nature of Deposit.	Associations.	Geological Age of Rock.	Eruptive Rocks in Vicinity.	Authority.	Remarks.
EUROPE.							
Spain.							
Mieres ...	Breccia of sandstone and schist.	Filling cracks and interstitial cavities; sometimes as impregnations.	Pyrites, mispickel and realgar.	Carboniferous.	.....	J. G. Klemm	.....
Almaden ...	Schists, limestone, quartzite and sandstone.	Veins or vein-like impregnations.	Native mercury, iron pyrites and chalcopyrite in small quantities; with quartz, a little heavy spar, and spots of bituminous matter.	Silurian...	Melaphyre	De Prado, Kuss, Becker, etc.	Mercury, 8·3 per cent. The veins follow, but sometimes cut across the stratification. The richness increases in depth.
Granada ...	Talcose schists	Veins	Grey copper, sulphide of nickel and cobalt, and oxides of iron.	Triassic ...	.....	Guillemin-Tarayre.	.....
Sierra de Montenegro.	Slates ...	Vein	.....	Silurian ?	.....	A. Heckmanns.	.....
La Creu ...	Sandstone ...	Veinlets...	Iron pyrites, quartz and carbonates.	.....	.....	G. F. Becker	.....
Germany.							
Rhenish Bavaria or Palatinate.	Limestone and sandstone.	Veins and impregnations	Iron pyrites, copper ores, lead and silver minerals (rare), calcite, quartz, chalcedony, heavy spar and bituminous matter.	Carboniferous.	Melaphyre	H. von Dechen.	Richest at top, gave out in depth. Cinnabar found replacing organic remains.

COMPARATIVE TABLE OF CINNABAR DEPOSITS.—*Continued.*

Locality.	Rock.	Nature of Deposit.	Associations.	Geological Age of Rock.	Eruptive Rocks in Vicinity.	Authority.	Remarks.
Austria.							
Idria	Schists, sandstones, dolomitic limestones and breccia.	In fissures, as bed-like veins, filling cracks, and as impregnations and stockworks.	Native mercury (small quantity), iron pyrites, quartz, calcite, dolomite, fluor (rare), idrialite, and anthracite in compact masses.	Triassic ...	Trachyte	M. von Lipold	Mercury 1-4 per cent. Fissures sometimes cut across and sometimes follow the planes of bedding. Richness increases in depth.
Horowitz (Bohemia).	Schists ...	In cracks at right angles to the bedding.	Iron pyrites, heavy spar and calcite.	.....	.....	Von Cotta...	Bed of hæmatite occurs here.
Thihuthal (Hungary).	Argillaceous schist.	Veins, in streaks and bunches.	Galena, blende (small quantity) calcite, dolomite, and country rock.	.....	Lava ...	Von Cotta...	Occurs at contact of dyke of lava with much-altered schists.
Italy.							
Agordo ...	Quartz-porphyry, sandstones, shales, graphitic slate, limestone and conglomerate.	Impregnations in porphyry and sandstone, and as stringers in shales. Great mass of it in the conglomerate as small grains and stringers.	Iron pyrites, mica, gypsum, calcite and quartz	Triassic ...	Porphyry	G. von Rath	0·2 to 1 per cent. of quicksilver ordinarily, but up to 24 per cent. At and near contact of porphyry and sedimentary rocks.
Mount Amiata	Limestones and siliceous beds.	Small irregular veins and impregnations in clay.	In veins with "neri"* and native quicksilver, in clay with iron pyrites, calcite and gypsum.	Eocene ...	Trachyte	B. Lotti, P. de Ferrari, and — Primat.	Found as masses and veinlets in clay with veins of calcite. Probably of post-Tertiary age.
Selvena ...	Marl	Impregnations	Calcite	.....	Trachyte	A. d'Achiardi	Calcite in stringers spotted with cinnabar.

* Sub-sulphide of mercury (Hg_2S).

COMPARATIVE TABLE OF CINNABAR DEPOSITS.—*Continued.*

Locality.	Rock.	Nature of Deposit.	Associations.	Geological Age of Rock.	Eruptive Rocks in Vicinity.	Authority.	Remarks.
Corsica.							
Cape Corso ...	Pegmatite, serpentine, euphotide schists and serpentiferous limestones.	Fissures in the rocks ...	Stibnite, iron pyrites, blende, sulphur and quartz.	.....	.....	D. Hollande	.....
Servia.							
Mount Avala	Serpentine (alteration product of an enstatite-olivine rock).	Vein-like, in seams and stringers of quartz and heavy spar which intersect the vein-matter in all directions; and as impregnations, and replacement (?).	Native mercury, calomel (in small quantities), chrome iron, iron pyrites, millerite, galena (rare), chalcedony, quartz, calcite, dolomite, barytes and iron oxides.	.....	.....	A. von Groddeck.	.....
Turkey in Europe.							
Serajevo ...	Schists and limestones.	Veins and beds ...	Iron pyrites, blende, traces of gold, calcite, dolomite, and country rock.	.....	.....	A. Conrad ...	Veins nearly vertical and sometimes several inches thick.
Russia.							
Bachmut ...	Sandstone overlain by clay-slate.	Impregnations and in cracks.	Galena	Carboniferous.	.....	Tschermak	.....
ASIA.							
Siberia.							
Nertschinsk ...	Limestone ...	Little veins and bunches	Calcite and quartz ...	.....	Granite ...	Von Koksharow.	.....
Japan.							
Hirado ...	Sandstone ...	Small fissures and seams	.....	Carboniferous.	.....	H. S. Monroe	.....

COMPARATIVE TABLE OF CINNABAR DEPOSITS.—*Continued.*

Locality.	Rock.	Nature of Deposit.	Associations.	Geological Age of Rock.	Eruptive Rocks in Vicinity.	Authority.	Remarks.
Borneo.							
Tagora ...	Metamorphosed shales and sandstones.	Irregular, and in vein-like developments.	Iron pyrites, stibnite (rare), calcite and heavy spar.	.....	.....	.....	In slate—more rarely in sandstone.
AFRICA.							
Algeria.							
Palestro ...	Limestone ...	Veins	Decomposed blende and galena.	Upper Cretaceous.	.....	A. Heckmanns.	Ores contain silver and mercury (sometimes 3½ per cent.).
NORTH AMERICA.							
California, U.S.	Slates (mostly metamorphosed) granite, ancient porphyry, andesites, rhyolite, basalt.	Impregnations, or as beds and deposits, in crevices, as stockworks "chambered veins" (Becker).	Native mercury (rare), iron pyrites, quartz, chalcedony, calcite, dolomite, pearl spar, chlorite and bituminous matter.	Cretaceous	Serpentine & trachyte.	G. F. Becker — Coignet.	Mercury 1–3 per cent. Probably deposited from hot sulphur springs in post-Pliocene times.
Mexico.							
Calzones ...	Pitchstone-porphyry.	Vein (S. Juan de la Chica).	Native mercury, etc. ...	.....	.....	A. von Humboldt.	Walls impregnated with cinnabar and quicksilver.
Durasno ...	Porphyry ...	Layers, resting on porphyry.	Native mercury, etc. ...	.....	.....	A. von Humboldt.	Surmounted by beds of shaly clay containing fossil wood and coal.
Guadalcázar	Limestone ...	Layers and stockworks.	Guadalcázarite, native sulphur, calcite and fluorspar.	Cretaceous?	Granite & porphyry.	Ramirez ...	Deposits generally separated from country by gypsum.
Guadalcázar	Granite ...	.....	Pyrites and galena ...	.....	.....	A. Nöggerath	.....

COMPARATIVE TABLE OF CINNABAR DEPOSITS.—*Continued.*

Locality.	Rock.	Nature of Deposit.	Associations.	Geological Age of Rock.	Eruptive Rocks in Vicinity.	Authority.	Remarks.
Huitzuco ...	Metamorphic slates and limestones.	Pockets, layers and veins	Stibnite, cinnabar, as pseudomorphs after stibnite, quartz, and gypsum.	.....	Serpentine, etc.	T. L. Laguerenne.	The ordinary ore is livingstonite.
Chilapa ...	Metamorphic slate.	Vein	Quartz and iron oxides	.....	.....	G. F. Becker	Fragments of country in it—ore impregnates entire width.
SOUTH AMERICA							
U.S. Colombia.							
Quindiú, Ibague.	Talco-chloritic, arenaceous and aluminous schists.	Impregnations and veinlets.	Iron pyrites, oxides of iron and manganese, quartz, calcite, clay, etc.	Silurian?	Diorite ...	.....	.....
Ecuador.							
Azogue ...	Ancient sandstones.	Veins	.....	.....	.....	G. F. Becker	.....
Peru.							
Chonta ...	Sandstone ...	Bed of clay, sand, iron pyrites and cinnabar.	Iron pyrites, clay and sand.	Early Palæozoic.	.....	Bugdoll ...	.....
Ancachs ...	.. ...	Veins	Galena, blende, iron pyrites, and grey copper.	.....	.....	M. du Châtenet.	.....
Yauli... ...	Schists and sandstones.	Seams and pockets in quartz-veins.	Iron pyrites	.....	.....	Bugdoll ...	Hot sulphur springs close by deposit sulphur at the surface.
Huancavelica (Santa Barbara mine).	Sandstone and clay-slate.	Bed-impregnations, stringers and stockworks.	Iron pyrites, mispickel, realgar, galena, calcite and barytes.	Jurassic (Crosnier).	Porphyries, trachytic, lavas and granite.	Humboldt, E. de Rivero.	Hot springs near.

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part IV.—Gold.

Free-milling and Grinding-milling.†

The cost of plant and of treatment of this character, in various parts of the world must now be considered, and to do so the author has drawn up Table I. hereto appended, which gives particulars of the plant and the approximate cost, as far as it can be estimated, of various well-known mills, gathered from such information as can be obtained on the subject.

Turning to Table I., it will be remarked that the cost of plant per stamp-head is least in Wales and America and greatest in Queensland, Australia, though it might cost a good deal more in out-of-the-way parts of Africa, South America, or Asia. It is stated that some of the earlier Johannesburg batteries cost £800 per stamp,‡ and the old Eureka mill, in California, cost £500 per stamp.

The low cost of the Welsh§ and American mills is partly explained by the fact that there are no heavy duties and comparatively slight freight charges to pay, which are a heavy item in other places, such as Queensland. The chief reasons, however, are that the former mills dispense with the grinding, tailings, and other plant that adds considerably to the cost of the Australian batteries, and that the prices of timber, lumber, and supplies are relatively very large in the Colonies.

For example:—Lumber in California costs about 75s. per 1,000 B.M., and in Queensland, Oregon pine ranging in size from 3 inches by 2 inches up to 9 inches by 9 inches, cost about 18s. 6d. to 21s. per 100 B.M. in 1887-8. Oregon lumber (imported) ranging in size from 8 inches by 8 inches up to 12 inches by 12 inches‖—cost about 1s. 6d. to 2s. 9d. per lineal foot in 1887-8. Redwood boards (well-seasoned) up to 18 inches in width, cost about 26s. to 28s. per 100 B.M. in 1887-8.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351, and vol. v., page 271.

† For the treatment of gold ores.

‡ *Mining Journal*, August 27, 1891, page 977.

§ The lower wages-rates, freight-rates, and price of materials in Wales explains the difference which exists between the Welsh and the American costs.

‖ In sticks, 36 to 40 feet in length.

TABLE I.

SHOWING THE CHARACTER AND APPROXIMATE COST OF FREE-MILLING AND GRINDING-MILLING PLANT IN DIFFERENT LOCALITIES.

Locality, District, and Name of Mill or Company.	Nature of Mill Site.	Character of Process.	Water or Steam Power.	Rock Breakers.	Stamps.	Weight of Stamps.	Automatic Feeders.	Number and Kind of Concentrators.	Pans.	Settlers.	Other Machinery.	Approximate Cost of Milling Plant and Mill Buildings.	Cost of Accessory Plant.	Cost per Stamp.	Remarks.
						Lbs.						£	£	£	
Wales—															
Dolgelly—															
New Morgan ..	*a*	Battery amalgamation and plates with ripples.	120 H.P. turbine & 125 H.P. eng.	2	40	850	8	Embrey pattern concentrs. used in old mill; not now employed.	....	..	..	8,500*b*	..	212	10 stamps in a battery, with pulley in the middle of the cam shaft. 4½ tons of coal burnt per day.
Italy—															
Pestarena—															
Pestarena	..	....	Water power..	1	..	*c*	..	Inclined tables	....	..	*d*	....	..	..	.....
India—															
Wynaad—															
Indian Consolidated (Phœnix)	*e*	Free gold in battery, and plates, ripples, and pans.	Turbine and engine.	..	20	850	..	4 end-blow tables.	7 Wheelers ..	6	*f*	....	..	..	There are six Wheelers and four Hungarian mills in reserve.
Transvaal (So. Africa),															
Johannesburg—															
Jumpers	*e*	Free gold in battery and plates outside.	Comp. engine, 100 H.P.	4	100	780	..	20 Frues, 4 shaking tables, 2 Buddles.	2 Wheelers, 1 Berdan.	1	*r*	45,395	..	454	*v*; *i* 240 × 60; 550 tons of coal burnt per month.
Dakota (U.S.A.)—															
Deadwood Gulch—															
Father de Smet ..	*g*	Inside & outside plates and mercury traps and sluice boxes.	Engine with Meyer cut-off (20 × 42).	4	80	758	16	Blankets ..	2 clean-up ..	1	*p*	20,338	653	254	*h*; *i* 68 × 60 *j*; 11 cords of wood per day.
Lead City—															
Highland	*g*	Do. ..	Eng. 350 H.P., Corliss.	6*k*	120	850	24	Blankets ..	2 clean-up ..	1	*p*	51,042	..	425	*l*; *i* 100 × 80; 14 cords of wood per day.
Homestake	*g*	Do. ..	Eng. 155 H.P., aut. cut-off.	5	80	850	16	Blankets ..	2 clean-up ..	1	*p*	....	..	..	*l*; *i* 100 × 80; 11 cords of wood per day.
Golden Star ..	*g*	Do. ..	Eng. 300 H.P. Harris-Corliss.	6	120	850	4	Blankets ..	2 clean-up ..	1	*p*	....	..	..	*l*; *i* 120 × 88¾; 14 cords of wood per day.

TABLE I.—*Continued.*

Locality, District, and Name of Mill or Company.	Nature of Mill Site.	Character of Process.	Water or Steam Power.	Rock Breakers.	Stamps.	Weight of Stamps.	Automatic Feeders.	Number and Kind of Concentrators.	Pans.	Settlers.	Other Machinery.	Approximate Cost of Milling Plant and Mill Buildings.	Cost of Accessory Plant.	Cost per Stamp.	Remarks.
						Lbs.						£	£	£	
Alaska (U.S.A.)— Douglas Island— Alaska-Treadwell ..	*g*	Inside & outside plates and mercury traps and sluice boxes.	6 ft. Knight and 6 ft. Pelton.	6	240	900	48	96 Frues ..	....	..	..	....	..	..	Concentrates chlorinated.
California (U.S.A.)— Plumas County— Plumas-Eureka ..	*m*	Inside & outside plates & silver sluices.	8 ft. Knight and 75 H.P. engine.	3	60	850	12	20 Hendys, 5 Pattons, 3 Duncans.	2 Patton pans, 1 clu.-up barrel, & 1 batea.	2	..	10,417	..	174	Tailings are ground in 30 arrastras and concentrates chlorinated.
Nevada County— North Star	*g*	Do. ..	6 ft. Pelton, 4 ft. do., & engine.	2	30	850	6	12 Triumphs ..	1 clean-up, 1 barl., 1 batea.	..	..	....	..	..	.....
North Queensland— Charters Towers— Day Dawn Blk. (Burdekin) ..	*m*	Free gold in battery and plates and pans.	100 H.P. eng.	3	60	800	12	24 Frues, 18 Br. & Stansfield.	2 Berdans, 24 Wheelers.	12	..	52,146*n*	..	869	.....
Day Dawn P.C. (Excelsior) ..	*e*	Do. ..	132 H.P. eng.	1	40	896	..	8 Brn. & Stans.	38 Ber, 16 Wh.	8	..	50,000*o*	..	1,250	.....
Bonnie Dundee (Plants)	*g*	Do. ..	45 H.P. eng.	..	15	896	..	6 Brn. & Stans.	1 Ber., 11 Wh.	7	..	16,000*o*	..	1,066	.....
Defiance	*e*	Do. ..	40 H.P. eng.	..	15*q*	800	..	2 Buddles ..	2 Wh., 24 Ber.	1	..	12,000*q*	..	800	With machinery since added, will probably come to about £1,100.
New Queen	*e*	Do. ..	68 H.P. en. with comp. cylrs. and var. expn. gear.	..	25	800	..	2 Buddles, 2 Br. & Stans., 5 end-blow tables.	39 Ber., 2 Wh..	3	..	20,000*o*	..	800	.....
Rishton— Disraeli	*m*	Do. .	50 H.P. engines	1	20	760	4	8 Frues ..	8 Ber., 4 Wh. *s*	2	..	20,000*ot*	1,753*u*	1,000	*v*; *i* 132 × 50*j*.
Etheridge— Durham and Lord Byron	..	....	82 H.P. engines	1	20	900	..	2 Frues, 4 Br. & Stansfield.	10 Berdans, 8 Wheelers.	2	..	35,000*o*	..	1,750	Freight and labour are much more expensive than in the Charters Towers district.
Cumberland ..	..	....	92 H.P. engines	..	30	900	..	4 Frues, 4 Br. & Stansfield.	3 Berdans, 20 Wheelers.	3	..	35,000*o*	..	1,166	

TABLE I.—*Continued.*

a Laid out in steps with considerable fall, involving about 2,000 cubic yards of cutting in decomposed granite.
b As stated by Messrs. Fraser & Chalmers, who supplied the machinery.
c One pair of rolls, 20 inches × 13 inches.
d Frankfort mill.
e Slight fall.
f 7 Hungarian mills.
g Slightly graded.
h Batteries face to face in two rows, with tables in middle.
i Dimensions of mill building.
j Exclusive of engine-room.
k No. 5 Blake.
l Batteries back to back in two rows.
m Graded.
n Report of the Company, July 22nd, 1890.
o Report Department of Mines, Queensland Government, 1889.
p Grizzlies, boilers, etc.
q Report Department of Mines, 1879. (5 stamps, 3 settlers, 2 Brown & Stansfield concentrators, 2 percussion tables, 24 Berdans, and 2 Wheelers have been added since.)
r 1 Tangye double 10 inches cylinder geared pump, capable of pumping 35,000 gallons per hour.
s Large wooden-sided combination pans, 5 feet diameter.
t This sum includes machinery, grading, timber, and erection of building, fittings and hardware, and freight and duty; also a full stock of tools and supplies for twelve months.
u Published Report of Mr. C. P. Purinton, August 15th, 1887.
v Batteries in line.

"Charters Towers," and "Rishton" are within the Charters Towers gold-field.

The figures in the column of "Cost per Stamp" go to show that the more modern English mills, like the New Queen, Disraeli, and Day Dawn block, have been built cheaper than the mills locally erected previously in the same district. The Burdekin mill, which is an extremely fine piece of work, was built by a Maryborough (Queensland) engineer.

Hardwood boards, of different widths and thicknesses, cost about 23s. to 25s. per 100 B.M. in 1887-8. Kauri pine flooring, T. G. boards of ordinary width, cost about 30s. to 32s. per 100 B.M. in 1887-8. Cord-wood costs 18s. to 22s. per cord.

According to the Queensland Government returns (from which the particulars given in Table I. are taken) another point to be noted is that the cost of some of the older Charters Towers batteries, like the Excelsior plant, seems to have been considerably in excess* of that of the newer mills, while even among these latter there are considerable variations of cost.

In this connexion the author would like to draw attention to several points which invariably affect the capital expenditure on any two different mills, and it is therefore of the first importance both to the mill-wright and the mill owner to make due allowance for them in any estimates made beforehand.

The prime cost of a mill, in fact, depends entirely on a variety of conditions:—

* This is no doubt partly owing to the higher cost of wages and freight in the early days of the field (everything having to be carted from Townsville on the coast), but, on the other hand, there was no duty to pay, and the price of bush timber was no doubt lower than it was five years ago.

(1) The size of the mill in regard to the tonnage of ore it is designed to handle.
(2) The general nature of the machinery and the mill site.
(3) The internal details of the building (in grouping and placing various machines together), and the choice of materials used in its construction.
(4) The distance the machinery and appliances composing the plant have to be transported from the place of manufacture, and the ease or difficulty of delivering them at the site of the works.
(5) The cost of the machinery at the foundry.
(6) The duties and commissions paid upon it.
(7) The local cost of labour and of timber, and other structural materials (purchased on the spot or elsewhere) delivered.
(8) The efficiency of the labour employed in their erection, and the time the works are under construction; which is contingent more or less on the willingness and ability of a company to provide for a heavy outlay covering a short period, or preferring payments distributed over a certain length of time.

To illustrate what is meant, the author will take two Queensland batteries with which he has been personally connected, the New Queen and the Disraeli.

The New Queen mill was designed and built on the best Australian plans, under the superintendence of Mr. Geo. Cavey; the erection of the machinery being entrusted to a first-rate English millwright and machinist, Mr. R. Robinson; and the battery has very justly earned the reputation of being one of the finest of its class on the Charters Towers field.

The Disraeli battery, on the other hand, was designed by Mr. J. Deby, of London, on American lines (the minor details and design of the building being left to the author's discretion), and it was erected under his supervision and that of his assistant, Mr. H. L. Lawrence; Mr. Wm. Reed, a most excellent English mechanic from the Sandycroft foundry (which supplied most of the machinery)* erecting everything in running order to plans under his orders.

Now, if we take the actual difference of cost per stamp in the New Queen mill and in the Disraeli, as shown by Table I., we find that the

* Messrs. Fraser & Chalmers furnished the Frue vanners and Messrs. Langland of Melbourne, the extra berdans.

latter plant (erected) apparently cost in proportion about £4,000 more than the New Queen battery, and it is a matter of interest to learn the cause of this discrepancy, of which the author can offer several explanations bearing on the points before mentioned.

Before referring to these, however, the author should point out that the figures given in Table I. as the cost of the Rishton mill are in reality fully £1,000 in excess of the actual sum expended, which should be £19,000, reducing the difference to £3,000. Further, the £19,000 in question includes the cost of a tram line, an expensive pumping station, and tailings-flume (the former to pump water from, and the latter to carry the tailings half-a-mile to, the river). These items do not enter into the cost of the Queen battery. Deducting their cost (according to figures given in a report by Mr. C. P. Purinton, published by the Company, Aug. 15th, 1887) we would have left (£19,000 — £1,605 18s. =) £17,394 2s., or an actual difference in cost of only £1,394 2s. between the two batteries. If, further, the cost of the reservoir in addition to the other outside works be subtracted, it would make the actual cost of the Rishton battery itself £17,246 13s. 6d., or £862 6s. 8d. per stamp, which is what doubtless it actually cost, viz., £8,146 spent on mill-building, construction, grading, and foundations, and the balance of £9,100 on machinery, freight, duty, etc. If a comparison were instituted on this basis it would make the difference in cost between the two mills only £62 6s. 8d. per stamp, or £1,246 13s. 6d. in total. We will assume, however, a supposed difference of £3,000 for the sake of illustrating a case in which this would be actually amply justified.

1. By the relative difference in the size of the two plants.
2. By the difference in the general nature of the machinery composing the two plants, and of their respective sites, which necessitated extra fittings and provisions, which could be dispensed with in the one case but not in the other. Such, for instance, as the cost of heavy framing and spacious bins,* required for the rock-breaker and automatic feeders of the Disraeli, and the extra strong framework needed for the combination-pans and large settlers, as compared with the small ordinary iron-wheelers, berdans, and settling-pans used at Charters Towers. To this may likewise be added

* In this connexion, it is a point worth noting that the planking of ore-bins are best laid on joists resting across the inclined timbers of the framework. The sheathing will then run lengthways down the bin and not across it; if double planked, the joints should be "broken" by covering the lower ones with the top boards.

the extra space taken up in the Rishton mill by the vanners,* which require a solidly and evenly boarded floor, and the fact that in a mill like the Disraeli, provision must of necessity be made to protect the superstructure as far as possible from the attacks of white ants, by bedding the mud-sills of the building on masonry, which is not such a necessary object where a rough stick can be put in or taken out without any great difficulty.

3. By the arrangement of the machinery and consequent difference of design of the internal details of the buildings. The Disraeli battery, being built in steps and storeys, requiring heavy grading, faced with retaining walls of suitable strength, and a massive framing of squared-timber† to support the load of the building and machinery, so as to provide sufficient fall for the ore to descend by gravity, with as little handling as possible, through the mill.

   The New Queen, on the other hand, like most of the other Charters Towers batteries (except the Burdekin and Plants), possessing such slight fall as to allow of the use of ordinary round timber (barked, but otherwise untrimmed) in the construction of the buildings, and requiring no expensive foundations, except for the mortar-blocks of the stamps themselves.

4. By extra freight; Rishton, where the Disraeli battery was located, being 22 miles from the railway terminus (which is at Charters Towers itself), so that an additional charge of £2 to £2 5s. a ton can be reckoned on all material used, representing a by no means insignificant items in the sum total.

Table II. gives the cost of treatment in a tabulated form at several of the mills, which have been referred to in Table I., and will be found useful for purposes of comparison.

* The cost of adding 24 vanners to the Montana Company's mill, at Marysville, is stated to have been £3,437 10s., shed and machinery included.

† Imported Oregon pine; as sticks of sufficient size and length (some running up to 12 inches by 12 inches by 41 feet) could not be obtained in the district.

TABLE II.

Showing the Cost of Free-milling and Grinding-milling Treatment in various Localities.

	Europe.			Asia.		South America.	Australia.						New Zealand.	South Africa.	
	Wales.		Italy.	Wynaad.	Mysore.	Venezuela.	Burdekin	Charters Towers.			Rishton.	Victoria.	The Thames.	Johannesberg.	
Name of mill	40-Stamp Mill. New Morgan		Pestarena.	20 Stamp. Indian Phœnix.	90 Stamp Mill.	60 Stamp. El Callao	60 Stamp. Day Dawn Block.	40 Stp. Day Dawn	50 Stp. P.C.	25 Stamp. New Queen.	20 Stamp. Disraeli.	60 Stamp Mill. S. Clunes United.	33 Stamp Mill. Saxon.	20 Stamp. May Consolidated.	100 Stmp. The Jumpers.
Tons crushed	1891. 960 (= 21,760 12 mos.)	1,700 (= 19,550 12 mos.)	Cost per ton of picking and crushing in stone breaker and rolls, 2s. 8d. Do. of amalgamation, including mercury, wear and tear, etc., 9s. 2d.	1884. 250 (= 12,574 12 mos.)	1892. 38,727	1891. 58,949	1890-91. 20,078	1889. 21,920	1890. 27,636	1889. 5,113 (= 10,226 12 mos.)	1887. 890 (= 10,680 12 mos.)	? 28,820	? 63	1889-90. 14,992	1889-90. *w* 24,795
Time crushing lasted	320 hours	627½ hrs.		6 days.	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.	6 months	1 month	12 mos.	24 hours.	12 mos.	6 mos.
Labour per ton crushed, cost	s. d. 0 8	s. d. 0 10½		s. d. 0 11½	s. d. ..	s. d. ..	s. d. 4 3½	s. d. 6 5	s. d. 5 5¾	s. d. 5 10	s. d. 4 9½	s. d. 0 10½	s. d. 2 10½	s. d.	s. d.
Material	0 2	0 6½		1 0½	..	..	0 11	2 0½	1 6¾	1 5	1 0	1 4½	?	13 0½	10 1
Fuel	..			Nil.	..	..	1 7½*m*	2 4*p*	2 3*p*	2 2½*p*	1 3¾*m*				
Repairs	?	?		0 8½	..	..	1 1½	0 6⅝	0 1	0 4½	0 1½		1 2½		
Transport	0 6½	0 7		0 5	..	..	2 2*n*	0 6½*q*	0 3½*r*	2 4*s*	0 6*t*	0 8¶	?	2 6¾	0 10½
Water	Nil.	Nil.		Nil.	..	..	Nil.	Nil.	Nil.	Nil.	1 5¼*u*	Nil.	Nil.	Nil.	Nil.
Total cost per ton, using as motive power: Water	*a*1 4½	..	*b*8 10⅜	*c*3 0¾	..	..	..	..	..	..	..	..	..	..	..
Steam	..	..	..	..	†14 6	*d*5 0	*f*10 1½	*g*11 10⅝	*h*9 7¾	*i*12 2	*j*9 2	2 11	4 1	*v*15 7½	*x*10 11½
Water and steam	..	*a*1 11½	..	..	..	..	..	..	..	..	..	..	..	..	..

TABLE II.– *Continued.*

	North America.						
	Alaska.	Dakota.				Colorado.	California.
Name of mill	240 Stp. Alaska-Treadwell.	80 Stamp Mill. Homestake ..		120 Stamp Mill. Golden Star.		75 Stamp Mill. Hidden Treasure.	40 Stamp Mill. Empire.
Tons crushed	1890-91. 220,686	1887-88. 96,790	1880 153,372	1880. 121,910	1887-88. 146,565	1891. 30,720	? 21,000
Time crushing lasted ..	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.
Labour per ton crushed, cost	s. d. 0 $9\frac{3}{4}$*k*	s. d. 1 $0\frac{1}{2}$	s. d. 2 $2\frac{3}{4}$	s. d. 1 $4\frac{1}{2}$	s. d. 0 $10\frac{3}{4}$	s. d. 1 7	s. d. 0 $10\frac{1}{4}$
Material	0 $6\frac{1}{4}$*l*	*0 $2\frac{1}{4}$	0 $8\frac{1}{2}$	0 $3\frac{3}{8}$	0 $2\frac{1}{4}$		0 $5\frac{3}{4}$
Fuel	0 $3\frac{3}{8}$	1 $1\frac{3}{4}$	1 $2\frac{3}{4}$	1 $0\frac{3}{8}$	1 $1\frac{3}{4}$		
Repairs	0 $1\frac{1}{2}$	0 $4\frac{1}{2}$	0 8	0 $1\frac{5}{8}$	0 $6\frac{1}{8}$	1 8	0 4
Transport	?	?	0 $1\frac{1}{4}$	?	?	?	?
Water	Nil.	0 $8\frac{1}{2}$	0 $5\frac{1}{2}$	0 $5\frac{1}{8}$	0 $8\frac{1}{2}$	?	Variable.
Total cost per ton, using as motive power — Water ..	*e*1 9	..	..	..	..	..	1 8‖
Steam ..	..	*y*3 $5\frac{1}{2}$	*z*5 $4\frac{3}{4}$	*z*3 $2\frac{3}{4}$	*y*3 $5\frac{3}{8}$	..	..
Water and steam ..	..	..	..	..	..	3 $3\frac{1}{4}$	..

a Manager's Reports, June to November, 1891.

b Estimate of "the agents."

c Report by the writer, August, 1884, published by the company, January, 1885. With a 20-stamp mill, possessing good fall, arranged with rock-breakers, stamps (with inside and outside "coppers"), and vanners, these costs could probably be reduced to 1s. 10d. per ton in India under like conditions.

d Report of Mr. George E. Webber, Jun., Superintendent, 1891. (6·29 fr. = 5s. per ton.) For the year 1891, 1,589 cords of wood were used, equivalent to 800 tons of coal of fair quality, costing £4 7s. per ton. In addition to crushing 58,949 tons of quartz, the engine did other work (pumping water and driving dynamo for hoisting-works, etc.), so only about three-quarters of the power employed was used for milling. Hence fuel used per ton of ore was equivalent to about 22 lbs. of good coal. The engine used at El Callao is a compound condensing tandem, supplied with steam by locomotive boilers with combustion chambers; steam pressure, 140 to 150 lbs. Mr. Hamilton Smith, Jun., (Notes on Gold Quartz Milling at El Callao, March, 1892) remarks, considering the high cost of labour supplies and fuel at El Callao:–"This is an exceedingly low rate. The above cost of 6·29 fr. per ton includes 0·60 fr. per share of general expenses. Such a charge is not generally made to milling accounts, so on the usual basis, the cost per ton for milling was 5·69 fr. or 4s. 6d."

e Company's Annual Statement, 1891.

f Company's Report, July 14th, 1891.

g Company's Report, 1889.

h Company's Report, 1890.

i Manager's Report, December 31st, 1889.

j Estimated by the writer. Mr. C. P. Purinton, who reported on the Disraeli in 1887, stated in his Report published by the Company, page 9, that "the cost of milling depends in a great measure on the quantity milled," adding:– "If the mine supplied ore enough in sufficient quantity to keep the mill constantly employed, the ore could be milled for not to exceed 5s. per ton, whereas the cost now is fully 10s." If general expenses and management (which for the sake of uniformity with the other mills is not included in the author's figures) be apportioned and added to his estimate, it tallies almost exactly with Mr. Purinton's. An extra charge per ton milled ought to be added to all estimates for the amortization of the capital, calculating it at from 10 to 15 per cent. of the gross cost of the plant, according to the probable life of the works. It is not unusual to write off 5 to 10 per cent. on machinery and $2\frac{1}{2}$ to 5 per cent. on buildings; but this proportion depends on circumstances, where (as in some cases, the materials of the building when "sold off" would command a better price comparatively speaking than the machinery.

TABLE II.—*Continued.*

ALASKA-TREADWELL.

k Subdivision of Labour—	Cost per Ton. Dol.
Foremen	0·0215
Amalgamators	0·0258
Feeders	0·0402
Oilers	0·0098
Concentrators	0·0333
Rock-breakers	0·0168
Do. (whites)	0·0343
Ditchmen	0·0090
Do. repairs	0·0033
Total	0·1940

l Subdivision of Material—	Cost per Ton Dol.
Shoes and dies	0·0608
Concentrator fittings	0·0078
Screens	0·0034
Rock-breaker supplies	0·0059
Feeder do.	0·0032
Miscellaneous	0·0032
Guide-blocks	0·0009
Oils and lubricants	0·0013
Lumber	0·0015
Rope and hose	0·0029
Water-wheel supplies	0·0019
Mortar and aprons	0·0082
Mercury	0·0088
Cam-shafts	0·0022
Battery linings	0·0029
Electric light	0·0062
Total	0·1211

m The Disraeli and, the writer believes, the Day Dawn Block use tubular boilers.

n Transport by rail (12 miles).

o In 1889, 10 stamps, 12 Berdans, and 2 concentrators were added to the plant (specified in Table I.) and came into operation the following year.

p The Day Dawn P.C. and New Queen use Cornish boilers, owing to the corroding action of the Charters Towers water. Fuel is also somewhat cheaper at Rishton and the Burdekin than in the Towers.

q Transport by rail (1½ miles). *r* Ditto (1½ miles). *s* Transport by cart (¾ to 6 miles).

t Transport by horse-tramway (½ mile).

u Labour, 10¾d.; fuel, 6½d. Pumping 150 gallons 2,500 feet, with a vertical lift of 108 feet.

v Report of the Company for year ending May 31st, 1890.

w Average duty of stamps, 2 tons 14 cwts. 3 qrs. per 24 hours.

x Report of the Directors for the half-year ending January 31st, 1890. The cost of milling during this period varied, it may be remarked, from month to month (from as low as 9s. 2d. per ton crushing 4,996 tons; to 22s. 8¾d. crushing only 1,245 tons).

y "Gold Milling in the Black Hills," by H. O. Hofman.—*Trans. Am. Inst. Min. Eng.*, vol. xvii., p. 498.

z Tenth Census of the United States, page 280.

* Includes supplies, candles, oil, mercury, lumber, and timber.

‡ Water power used for four months, steam for four months, and both water and steam for remainder of year.

‖ This figure is an average, the cost varying from 1s. 5½d. to 1s. 10½d. When steam is used 5d. extra must be added to the total given.

¶ Tramming the stone one-third mile and breaking it on contract.

† Of this sum 5s. to 6s. 6d. is probably chargeable to treatment of tailings by grinding.

NOTE.—To institute a fair comparison between the total cost at the different mills recorded in the table, the cost of transport must, of course, be deducted from the total cost, making allowance as well for difference in the relative quantity crushed in each case, etc.

The cost of treatment in different localities, using the same milling process, is quite as variable as the cost of plant.

The author does not think he is overstating the fact in saying that at most of the older Charters Towers batteries (with 15 to 20 heads) the cost of milling runs from 14s. to 18s. per ton and more, including only such items as are given in Table II.

At Plant's mill, according to the report of the company (*Northern Miner*, September 19th, 1889), the profit for the year ending December 31st, 1888, was about £3,000, and if the stone milled be taken at 13,500 tons and the charge for crushing, including grinding, is assumed to have

TABLE III.

SHOWING RULING RATES OF WAGES IN DIFFERENT LOCALITIES AND STAFF OF VARIOUS MILLS.

Occupation.	20 Stamp Mill, India.			80 Stamp Mill, Dakota.*			120 Stamp Mill, Dakota.†			40 Stamp Mill, Wales.‡	
	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	Length of Shift.	Wages per Shift.
		Hrs.	s. d.		Hrs.	s. d.		Hrs.	s. d.	Hrs.	s. d.
Foremen	1	...	...	$\frac{3}{3}$	...	27 1	$\frac{2}{3}$	...	27 1	...	...
Amalgamators ...	2	12	3 4	1	10	16 8	1	10	16 8	12	6 6
Assistant do. ...	1	12	1 $5\frac{1}{2}$	4	12	14 7	4	12	14 7	12	5 6
Rock-breakermen	...	...	...	5	10	12 6	6	10	12 6	12	4 0
Feeders & stone-breakers ...	14	12	0 $8\frac{1}{2}$	2	12	12 6	4	12	12 6	12	4 0
Oilers	1	12	0 $8\frac{1}{2}$	2	12	12 6	2	12	12 6	...	...
Machinists ...	..	...	3 4	1	10	17 $8\frac{1}{2}$	1	10	17 $8\frac{1}{2}$	...	...
Pipe-fitters ...	...	...	...	$\frac{1}{3}$	10	14 7	$\frac{1}{3}$	10	14 7	12	4 6
Engine-drivers ...	...	...	3 $5\frac{3}{4}$	2	12	14 7	2	12	14 7	...	...
Firemen ...	...	...	...	2	12	12 6	2	12	12 6	...	...
Blacksmiths ...	1	12	1 8	...	...	...	...	...	...	...	...
Strikers	1	12	0 $7\frac{3}{4}$	...	...	...	...	...	...	...	...
Carpenters ...	1	12	1 8	...	...	...	...	...	...	...	...
Labourers ...	2	12	0 $8\frac{1}{4}$	...	...	...	...	...	...	8	3 6
Watchmen, etc....	4	12	0 $8\frac{1}{4}$	$\frac{1}{2}$	12	12 6	$\frac{1}{2}$	12	12 6	...	...
Panmen & oilers	...	...	...	...	...	...	...	...	...	...	...
Vanner-attendants	...	...	...	...	...	...	...	...	...	...	...
Total number	28	...	...	$20\frac{1}{2}$	...	...	$23\frac{1}{2}$	...	...	...	...

* Homestake. † Golden Star. ‡ Morgan G.M.C.

been somewhere about the usual price of 18s. to 22s. per ton,* then taking it at 19s. per ton, it leaves a balance of £9,825 for milling expenses during the year, which, divided by the gross tonnage crushed, would represent a milling cost of about 14s. $6\frac{3}{4}$d. per ton. This is a case of special interest, because wheelers are used entirely for grinding and amalgamating, to the exclusion of berdans, in which respect this mill differs from most of the others in the district. An examination of Table I. shows, however, that the cost of milling in the same locality varies greatly, depending upon:—

1. The general design and internal arrangements of the mill-building, for economizing and facilitating labour, and simplifying the plant, depending to a great extent on the selection of a suitable site and its proper utilization.
2. The general nature of the process in regard to the saving or loss of gold and mineral, which the disposition and character of the machinery effects.

* Depending on the amount of pyrites in the stone, and the time taken to grind a given tonnage (varying with the capacity of the pan).

TABLE III.—*Continued.*

Occupation.	20 Stamp Mill, Disraeli. Rishton.			Charter's Towers.					
	No. of Men.	Length of Shift.	Wages per Shift.	A 10 Stamp-battery	A 15 Stamp-battery	A 20 Stamp Mill.	A 25 Stamp Mill.	Length of Shift.	Wages per Shift.
				No. of Men.	No. of Men.	No. of Men.	No. of Men.	Hrs.	s. d.
Foremen	...	...	Rates much the same as at Charters Towers.	...	..	...	...	...	...
Amalgamators ...	1	12		2	2	2	2	12	16 8
Assistant do. ...	2	8		...	...	...	...	8	13 4
Rock-breakermen	1	8		...	...	...	...	8	10 0
Feeders & stone-breakers ...	1	8		3	4	5	6	8	10 0
Oilers	...	...		...	...	...	...	...	...
Machinists ...	½	...		1	1	1	1	12	16 8 18 4
Pipe-fitters ...	...	...		...	...	...	...	...	...
Engine-drivers ...	3	8		3	3	3	3	8 12	11 8 13 4
Firemen ...	...	...		...	...	...	...	...	...
Blacksmiths ...	...	...		...	...	...	...	8	13 4 16 8
Strikers	...	...		...	...	...	...	8	10 0 11 8
Carpenters ...	...	...		...	...	...	...	8	15 0
Labourers ...	...	...		...	...	...	...	8	10 0
Watchmen, etc....	...	...		...	...	...	...	8 12	10 0 11 8
Panmen & oilers	2	...		1 to 2	2	2	3	8	10 0 11 8
Tailingsmen and Vanner attend'ts.	1	..		*1	*1	*2	*2	12	11 8
Total number	11½	...	...	11 to 12	13	15	17	...	...

* Tailingsmen.

3. The gross tonnage that is handled by the mill in a given time (sometimes affected by climate).
4. The efficiency of the labour employed, and its cost.
5. The quality and the price of supplies, fuel, etc., used.
6. The power employed, and its method of application.
7. The situation of the works as regards water-supply, transport of ore from the mine to the mill, and disposal of tailings.
8. The efficiency of the general management.

All these points have to be studied from the standpoint of relative utility and comparative cost.

As regards the author's first proposition, effect of general design, nothing could illustrate what is meant better, than the saving of labour and material (as shown by Table I.) effected in mills, laid out like the Day Dawn block and Disraeli, as compared with batteries of the New Queen and Day Dawn P.C. type.

TABLE III.—*Continued.*

Occupation.	60 Stamp Mill, Dakota, California.			40 Stamp Mill, California Empire.			75 Stamp Mill, Colorado, Hidden Treasure.			35 Stamp Mill, New Zealand, Saxon.		
	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.
		Hrs.	s. d.		Hrs.	s. d.		Hrs.	s. d.		Hrs.	s. d.
Foremen... ...	1	...	27 1	...	...	...	...	...	...	...	...	...
Amalgamators ...	2	12	15 7½	2	12	12 6	1	12	24 3½	3	8	8 0
Assistant do. ...	...	...	...	...	...	...	1	12	13 10½	...	...	...
Rock-breakermen	1	10	12 6	1	12	10 5	...	...	...	...	...	...
Feeders & stone-breakers ...	2	12	13 6½	...	...	...	6	12	12 6	3*	8	6 8
										3†	8	5 0
Oilers	...	...	...	...	...	...	...	...	...	...	...	...
Machinists ...	1	12	18 9	...	...	...	...	...	...	...	...	...
Pipe-fitters ...	...	...	...	...	...	...	...	...	...	...	...	...
Engine-drivers ...	2	12	14 7	...	...	...	...	...	...	...	...	...
Firemen	2	12	13 6½	...	...	...	...	...	...	...	...	...
Blacksmiths ...	...	...	...	...	...	...	...	...	...	...	...	...
Strikers	...	...	...	...	...	...	...	...	...	...	...	...
Carpenters ...	...	...	...	...	...	...	...	...	...	...	...	...
Labourers ...	1	10	10 5	2	12	10 5	...	...	...	...	...	...
Watchmen, etc....	...	...	...	...	...	...	...	...	...	...	...	...
Panmen & oilers	...	...	...	...	...	...	...	...	...	...	...	...
Tailingsmen and vanner-attendants	...	...	...	1	12	12 6	2	12	12 6	3	8	3 4
Total Number	12	...	...	6	...	...	10	...	...	12	...	...

* Men. † Boys.

Wages in Wales run from 21s. to 39s. per week. The staff is composed of 7 men by day and 3 by night at the Morgan mill.

Wages in Italy run from 9½d. to 3s. 8d. per day at Pestarena. Girls earn 9½d. to 11d. per day; millmen, 2s. per day; labourers, 1s. 6½d. to 1s. 9d. per day; Smiths, 2s. 5½d. to 3s. 8d. per day.

Wages in South Africa are as follows:—Amalgamators (European), £20; assistant amalgamators, £15; rock-breakermen, £17 10s.; machinists, £30; engine-drivers, £26; blacksmiths, £30; carpenters, £22 10s.; and labourers, £13 per month. Natives earn £3 per month. The Jumpers mill employs a staff of 13 Europeans and 65 Kaffirs, with 4 European vanner-attendants and 5 Kaffirs in the vanner-house.

As both the Day Dawn batteries are first-rate mills of their class, and are in charge of first-class Queensland mill-men, and both the New Queen and Disraeli were under the writer's management, when the figures given in Table II. were compiled, he has no hesitation in taking these four plants for comparison. In regard to the New Queen, it will be noticed that the charge for labour, material, and repairs amounted to 7s. 7½d. per ton in 1889, as compared with 5s. 11d. at the Disraeli in 1887, a difference in favour of the latter of 1s. 8½d. per ton, notwithstanding the extra charges on supplies, etc., involved in transporting goods of every sort, an additional 22 miles to Rishton. What lends these two cases special value is the fact that, for all practical purposes, the 3rd, 4th, 6th, and 8th of the above considerations may be considered practically the same in both. The 1st, 2nd, 5th, and 7th are therefore evidently what really affect the point under consideration.

The nature of the plant affecting as it does the cost of treatment, must be always recognised in considering the cost of a mill. Now, if we assume, for instance, an equal tonnage crushed in the New Queen and Disraeli batteries, amounting to 10,000 to 12,000 tons per annum (running the same number of stamps) it is evident that there is a gross saving of between at least (1s. 8½d. × 10,000 =) £854 3s. 4d. to (1s. 8½d. × 12,000 =) £1,025 annually in favour of the Rishton works. Against this, strictly speaking, however, one must place the interest (reckoned say at 4 per cent.) on the extra capital outlay expended on the latter mill which, if we include accessory plant (such as reservoir pumping-station and race) is assumed to amount as before stated to about £3,000.

It will be noted that if one were to take the Burdekin or Excelsior batteries for comparison, from the costs given in Table I., the difference in the former case on 20 stamps erected would be £2,620 against the Disraeli. This, however, is easily explained by the difference in freight rates and the proportionately lower cost of constructing a large mill as compared with a small one; and as a great deal of the machinery was got from Maryboro' it had not to pay duty. In the case of the Excelsior mill it would be (20 × £250 =) £5,000 in favour of the Rishton mill.

It is indeed more than probable that, given the same mill-site, if the Disraeli battery had been erected at Charters Towers, it would have cost but little more than the New Queen per stamp-head, but taking the assumed capital expenditure in the two cases, and deducting interest as before remarked on £3,000 at 4 per cent., or £120 on the most unfavourable supposition; we might fairly attribute a net economy per annum of (£854 3s. 4d. — £120 =) £734 to (£1,025 — £120 =) £905 to be credited simply to the general design of the building and character and disposition of the machinery, which would repay the assumed extra capital cost with interest in between 3 and 4 years. What applies in this case evidently applies equally to any other similar 20 stamp mill in Queensland, entirely apart from the extra saving of gold a properly arranged mill is likely to effect; a matter which, though intimately connected with the question, comes under the author's second heading.

The saving that may be effected by arrangement of plant would, however, be far more striking if one had for comparison a 40 stamp mill, built like the Disraeli with a 40 stamp mill of the ordinary Charters Towers type, since the former could be run with 4 or 5 extra

men making say 16 all told, whilst the latter would require with the most careful management at least 24; while for an 80 stamp mill the numbers would stand relatively about as 26 to 44.

Reckoning mill wages on the average at 10s. 7d. per day (shift) and the working year at 303 days, this represents a saving on wages alone, running a 40 head battery, of £4 4s. 8d. per diem or £1,282 14s. per annum, or on an 80 stamp mill £2,886 1s. 6d. per annum, *plus* the extra saving on supplies, repairs, etc., which reckoned at only 8d. per ton would amount to a considerable sum, say £666 to £1,333 additional.*

The first cost of machinery is often only a small part of the total cost of erection, and it is therefore of the first importance that the best design and execution should be insisted on.†

Again, Messrs. McDermott and Duffield cite the case of two gold mills in Venezuela, running side by side, and owned by the same company, which well illustrates this. Both are 66 stamp mills operated on the same ore. The one poorly designed and built, the other embodying the results of practical mill-men's experience. The one mill can be made to average 93 tons crushed daily. The other averages over 143 tons. The first costs in working expenses, 18s. 9d. per ton of ore, the latter only 6s. 3d. These mills are ordinary gold mills, and it will appear incredible to those who have not run such machinery that such differences can exist, believing that a stamp mill is merely a medium for crushing, and that economy always consists in purchasing it in the cheapest market and erecting it anyhow.

The author's second proposition, viz., the saving or loss of gold, the details of the arrangement of a mill effects is one that is exceedingly complex, as there is an infinite choice of machinery and method of

* As bearing on this question the report of the Superintendent of the El Callao, 1891, may be quoted:—"Notwithstanding the increased amount of work the mill has had to perform and its growing age, few renewals and repairs have been required, and at the close of the year all its parts remain in good running condition."

† Mr. Hamilton Smith, Jun., says: "The old El Callao mill for the year 1882 crushed 22,405 tons of quartz at a cost of 78·30 francs per ton. A comparison of these results with the results (see Table II.) obtained by the new mill is very instructive to mining men; as there could be no better proof of the advantages of first class mining machinery coupled with judicous management. Had it not been for the construction of this new mill, El Callao mine would have suspended operations years ago." The cost for the year ending December 31st, 1892, was 6·22 francs per ton, whilst the average assay of the tailings was 3½ dwts. per ton.

grouping it, which more or less affect the question, depending on the nature of the ore; various illustrations of this will, however, be given later on.

It is a matter which must be confided to the technical knowledge and experience of the engineer who designs the plant presumably with certain specified objects in view, a number of which have been incidentally noticed. The losses in gold-milling are, like local costs, extremely variable.

In Gilpin County, Colorado, for example, 113,427 tons were milled during the census year ending May 31st, 1880, out of which :—

Per Cent.			
65	of the gold was saved by	direct amalgamation.	
4	„	„	pan treatment
7·05	„	„	re-concentration of tailings.
Total, 76·05	The ore running £2 1s. 3½d. in gold and 1s. 3d. in silver.		

The above agrees with the writer's experience in India, where he found 76·2 per cent. was saved by direct amalgamation and pan treatment.

The latest recorded results of one of the best mills of Gilpin County, which is said to be fairly representative of present Colorado practice, shows that treating an ore containing 7·46 ozs. of gold and 32·86 ozs. of silver, it is possible to extract by milling 93·8 per cent. of the gold and 74 per cent. of the silver (including the value in the concentrates) out of which 70·4 per cent. of the gold and 42·6 per cent. of the silver was extracted by direct amalgamation. This, the writer thinks, shows, that whilst the saving by direct amalgamation in Colorado remains much the same as it was, a steady improvement has taken place in the manipulation of the pyrites.

Treating 25 dwts. ore at El Callao the loss is said to be $3\frac{1}{2}$ dwts.; in 1891 it was $2\frac{3}{4}$ dwts. In Dakota, the Homestake and Golden Star mills claim to save 85 per cent. of the free gold, $\frac{1}{2}$ to $\frac{3}{4}$ dwt. being unrecovered in the tailings. The tailings of the Alaska Gold-mining Company run $\frac{1}{2}$ to 1 dwt.

Mr. G. T. Deetken, some years ago, made some elaborate experiments at one of the best Californian gold mills of the day, and found that 27 per cent.* was lost in the tailings giving an extraction of 73 per cent. At the present time, with improved methods of treatment, the extraction

* This, again, about corresponds, the author believes, with the average loss (gross) in the Charters Towers mill tailings, crushing and treating 25 dwts. ore; it may be assumed that in concentrating and grinding the pyrites, the loss runs from 5 to 7 dwts.

in California probably approaches nearer 80 to 85 per cent., and one instance might be cited where it was as high as 82 to 94 per cent., the concentrates being treated by chlorination.

As most gold ores contain some gold combined with pyrites, in all probability, the average extraction of most mills treating stone of this kind often falls short of 75 per cent., and is occasionally as low as 45 per cent. if the stone contains much pyrites, which is dealt with in a crude manner.

In Victoria, treating $5\frac{1}{4}$ dwts. to $5\frac{3}{4}$ dwts. ore, the average extraction at the present day in the best mills is about 84 per cent., and has run as high as 87·6 per cent., which is the best record grinding milling can show under the most favourable circumstances with an ore exceptionally free. The milling practice of the district has steadily improved since 1861, when there was actually a loss of nearly 50 per cent. of the gold in the stone.

Though miners, as a class, possess just as much honesty and hard common-sense as people who follow many other callings, when it comes to consideration of these mill losses (although they can only be remedied by knowing exactly where the loss occurs and what it amounts to) the behaviour of a certain section of mining people and mill-men when confronted with facts of this sort reminds one forcibly of the behaviour of the ostrich confronted with death. The bird, it is said, sticks his head into the desert sands, and affects to disbelieve in the possibility of loss of life. The unpractical individual who generally loves to self-style himself the practical man* (who don't believe in assays, underground-surveys, and that kind of thing!) buries his eyes in the mill-sands, declares, believing it or not, that there is no loss of gold, and the facts, whether, as in some cases, from honest conviction, in others from dishonest expediency, become falsified.

It is curious that shareholders in mines seem actually to prefer to be deceived in this respect, and the mill-man who tells them that there is a loss in their tailings is lucky if he escapes being abused as well as thought a fool, simply because his neighbours choose to declare that they are losing nothing!

In this connexion, automatic samplers may be used with advantage to sample the crushed ore, pulp, and tailings regularly.

In August, 1884, the author put through a test crushing (in India) of

* A very different individual to the genuine miner or mill-man, of whom the late Mr. George Langtry, of Comstock fame, might be taken as a real type. See "Amalgamation on the Comstock Lode, Nevada," by Mr. A. D. Hodges, *Trans. Am. Inst. Min. Eng.*, vol. xix., page 216.

280 tons of stone, the particulars of which (given below) may be of some interest. It may serve as an illustration of the kind of information which ought to be obtainable of the working of every gold mill, and shows how, if the ore is regularly assayed (as it ought to be constantly), the results can be checked.

Number of stamps, 20; weight of stamps, 850 lbs.; lift, 9 inches; speed, 58 drops per minute.
Duty of battery, 41·3 tons per diem, or 2 tons 145 lbs. per stamp.
Order of drop, 3, 1, and 5 together, and 2 and 4 together.
Water supplied to each battery of 5 stamps, 3·52 cubic feet, or 22 gallons per minute.
Water supplied to other parts of the mill, 16 gallons per minute.
Ore per cubic foot of water discharged from the battery, $\frac{1}{3\cdot6}\frac{1}{3}$ cubic feet.
Mesh of screens, 196 holes per square inch (about $\frac{1}{3\frac{1}{2}}$ inch).
Wear of shoes, $1\frac{1}{2\cdot8}$ lbs. per ton of ore crushed.
Wear of dies, about half that of the shoes.
Amount of pyrites in the ore, $2\frac{1}{2}$ per cent.
Degree of concentration, 50 per cent.
Concentrates obtained from percussion-tables, 14 tons.
Yield of milled gold, 2 dwts. 10·7 grains.
Bullion fineness, 911·5
Tons of ore crushed, 280.
Time crushing lasted, 162 hours 35 minutes.
Mercury required to charge entire mill, 1,458 lbs.
Loss of mercury (owing to an accident at one of the mills this cannot be stated exactly).

	By Assay.	By Estimate.	Actually Obtained or Accounted for.
	Ozs. dwts. grs.	Ozs. dwts. grs.	Ozs. dwts. grs.
280 tons of ore, assaying 3 dwts. 5 grs. per ton contain	44 18 8	...	...
Gold obtained from stamp-boxes	...	...	2 12 0
Gold in the ore leaving the battery (deducting gold obtained from boxes)	...	42 6 8	...
Gold obtained from copper-plates	...	...	18 10 0
Gold in the ore after passing the plates (deducting gold obtained from plates and boxes)	...	23 16 8	...
Gold obtained from riffles	...	...	4 0 0
Gold in the ore after passing the riffles ...	*a*18 14 12	19 16 8	...
Gold obtained from the wheeler-pans ...	...	...	6 10 0*b*
Gold in the concentrated slimes leaving the wheeler-pans	12 5 0	12 14 12	...
Gold obtained from the Hungarian mills ...	...	...	2 13 0*b*
Gold in 9½ tons of settlings, collected in catch-pits	7 8 1	...	7 8 1*b*
Gold in the tailings of the percussion-tables	1 12 0	1 11 20	1 11 20*b*
Gold in the tailings of the settling-tanks, and unaccounted for	...	1 13 11	1 13 11*b*
			44 18 8

The sum of the results marked *b* (19 ozs. 16 dwts. 8 grains) should equal the assay marked *a* (18 ozs. 14 dwts. 12 grains) to be theoretically absolutely accurate.

No efficient check can be kept on the work going on at a mine or mill without a proper record of the working of each department being kept by its immediate heads, and a subdivision of accounts, which shows the cost per ton, under different general heads, in detail from month to month.

The author's third proposition (the influence on cost of the gross-tonnage handled in a mill in a given time) is instanced by the results of milling at the Day Dawn P.C. mill in 1890 as compared with 1889, the effect of milling 1,140 tons extra in the latter year being to reduce the expense on each item of cost (Table II.).

Referring to the capital charges in 1890, the directors in their report make the following remarks:—"The capital expenditure in question has been fully justified, for it has already greatly reduced the working charges, as will be seen from the following table:—

Year.	No. of Tons Treated, excluding Mundic Stone.*	Total Cost of Reduction and Extraction of Gold.†	Cost per Ton of Ore Treated.‡
		£ s. d.	£ s. d.
1889	26,551	45,896 12 10	1 14 7
1890	28,879	42,924 16 7	1 9 9
	+ 2,328	− 2,971 16 3	− 0 4 10

showing that although 2,328 more tons were treated in 1890 than in 1889 the cost was £2,971 16s. 3d. less." This statement should be obviously reversed; the capital expended having increased the output in 1890 being the cause of the reduced cost.

Take once more, for example, the New Queen. The author has shown in Table II. that the average cost of treatment in 1889 was 12s. 2d. per ton, but as was remarked in the report published from which these figures were taken—from September to November (of the year in

* Including ore crushed at outside mills (the amount crushed at the company's own mill being given in Table II.).

† Including mine expenditure, mill expenditure, expenditure on Rose of England lease, crushing ore at outside mills, and bullion-remittance expenses account.

‡ The cost, including general expenses at Charters Towers, appears to have been £1 16s. 1¼d. in 1889 and £1 10s. 9¾d. in 1890.

question) the actual cost ranged from 9s. to 10s. 4d. per ton, according to the quantity of stone crushed, which varied from 1,162 tons to 943 tons per month.

The effect of the quantity milled on the cost of milling is again evident if we compare the cost of milling at the Homestake and Golden Star mines in 1880 (Table II.), and the cost of milling in the new 60 stamp mill* of the El Callao Company, during several different years, tabulated below :—

Year.	Tons Crushed.	Cost per Ton. Francs.
1888	15,692	18·40
1889	43,629	12·60
1890	53,977	7·82
1891	58,949	6·29

The reason for this reduction is evident : with an enlarged scale of treatment, while the standing charges for management, skilled labour, etc., remain much the same, the amount of even the extra manual labour required is not by any means always increased in direct proportion to the additional tonnage crushed. It is to be recollected, however, that in crushing an extra quantity of stone its general average yield is likely to drop, as it often pays to crush stone in a large mill, which, with a smaller plant, would have to be discarded.

The yield of the Jumpers Company, for instance, crushing for six months with 30 stamps, fell from $18\frac{1}{2}$ dwts., to between $10\frac{3}{4}$ and $11\frac{3}{4}$ dwts., with 70 stamps; and adding 30 stamps to the Robinson 10 stamp mill, the average grade of the ore fell 50 per cent.†

The Charters Towers gold-field, again up to 1889, is reputed to have produced 1,915,051 ozs. of gold representing a yield of 1 oz. 10 dwts. 9 grains per ton ; in 1890, 121,406 tons 8 cwts. 2 qrs. of quartz were crushed for an average yield of 1 oz. 6 dwts. 19 grains ; and 174,000 tons in 1891, which only averaged 1 oz. 5 dwts. 9 grains.

Allowance must consequently be made for this decreased yield per ton in estimating the probable profits of a large plant when calculations are made, based upon the yield of an existing small mill.

Another point, however, in favour of large mills is that labour can be more easily specialized, which tends to reduce cost by promoting increased manual dexterity in individual operations, whilst by running reduction works up to their full capacity (a thing to be always aimed at) the fractional excess of unproductive labour, or, in other words, the time

* Supplied in 1885 by Messrs. Fraser and Chalmers.

† Messrs. McDermott and Duffield, page 5.

thrown away that has to be paid for, to perform duties which do not occupy one or more men a full shift, is reduced to a minimum.

The author's fourth proposition (the influence of the efficiency of the labour employed on the cost of milling) has a most important bearing on the question. It has been admirably dealt with by Mr. W. M. Howe in "Notes on the Bessemer Process."* Though speaking of the manufacture of steel, his remarks, which are quoted with some slight adaptations, apply to all reduction works: "A skilled manager, amalgamators and engine-drivers you must have, be the output large or small, whether the works run continuously or but a certain part of the time. The difference is that in the case of works with a small output, much of the time of these men, which you must pay for in full whether you use it or not, is either wasted or devoted to work, which in the case of a large output is performed by less intelligent, less skilled, less costly men; double your output and you scarcely need more of these highly skilled men. Most of the additional men are less skilled, many indeed are but assistants of the skilled nucleus which is necessary, and nearly as large in the case of the small as in that of large outputs, and though the foreman or mechanic who is to direct others, must add executive ability to the qualifications he would otherwise need and hence commands higher pay, yet the extra expense thus caused should raise the average cost of the daily wages, much less than it is lowered by the employment of the larger number of relatively unskilled assistants, due to the greater output. There are many trivial and very simple duties which call for little intelligence. With a large output those of each kind recur so often that cheap men can be fully occupied with them. The expensive men in cases of small output, do these trivial, simple acts with their costly labour, which in cases of large output is restricted to the difficult tasks which require it and which fully occupy it. Or, if justified by the greater output on the unit of which their pay forms a relatively small charge, you employ additional skilled and costly men, they are not mere duplicates of those you had before, they bring a different skill and additional knowledge to your aid, permitting economies and devising improvements otherwise unattainable."

This question of skilled labour, it will be noticed, is intimately connected with the production of the mine and capacity of the reduction works.

A large company, with perhaps several hundred men in its employ, requires the highest skilled organization to make it pay interest on

* *Trans. Am. Inst. Min. Eng.*, vol. xix., page 1,120.

a large capital, but the direct opposite may be said of a mine in its infancy, working on a small scale, with a small production, which generally means a small net profit. A staff of highly skilled and consequently highly paid officials would obviously be its ruin. It is therefore, fortunate that it is only when a mine reaches a certain size and depth, with a corresponding output, that the services of a skilled staff become indispensable to commercial profit on a large scale. There is a miners' saying that tributers will make a living where a company will starve, and there is a great deal of truth in it, if only because tributers have no general expenses to bear.

In the mining and milling of gold and silver ores, speaking from the writer's own experience, the skilled Anglo-Saxon workman, whether he be British, American, or Colonial, distances all competitors. Each subdivision of the race numbers in its ranks hundreds of miners (using the expression as covering mill-work as well) second to none in the world, though not a cheap man certainly. The payment of a low rate of wage to the skilled workman is by no means the only, or indeed the most expedient means of lowering the cost of production, since the cost of a day's labour in any responsible occupation is in reality in inverse proportion to the intelligence as well as the sinew and endurance it represents, and in reducing the rate of the one you run great risk of lowering the standard of the other, so long as superior skill can command better wages elsewhere.

The writer's contention is that the cheapest skilled labour is not always, indeed is very rarely, the most economical, if only because the highly-paid man is the most contented, better fed, and better housed man, and the author proposes to show that it is quite possible to cheapen costs without lowering wages, providing a reduction in working expenses is not demanded suddenly, whether by strikes or market fluctuations. To illustrate this,* the Atlantic mine, in Michigan, which produced native copper from 1873 to 1877, scarcely made ends meet, and had to levy calls ; since then, with the price of copper steadily declining, and also the yield of the ore, the cost of mining has been so greatly reduced that the company has made a profit every year. The workmen, who earned on an average from £10 8s. 4d. (50 dollars) to £12 5s. 10d. (59 dollars) a month in 1873, continue to earn almost as many dollars, and owing to the greatly reduced cost of living they are now very much better off and save really more money than they did twenty years ago.

* These particulars are taken from a leader on "The Free Coinage Question," in the *Engineering and Mining Journal* (New York), vol. lii., page 497.

The chief items of cost at the Atlantic mine for three years are shown in the following table:—

	1873.	1880.	1890.
Tons treated	51,048	169,825	278,300
Pounds of ingot produced	863,366	2,423,225	3,619,972
Percentage of yield	0·81	0·71	0·65
Average selling price, cents. per lb. ...	26·66	20·55	15·21
Cost of stoping per fathom	$22·28½	$14·35	$4·21
„ driving per foot	19·09½	12·90	5·10
„ sinking shafts per foot	38·03	18·04	22·90
Cost per ton, mining and all surface expenses, taxes, etc.	5·12	2·14	1·04
Cost of transport, 3 miles from mine to mill, per ton	0·15	0·09½	0·03½
Cost of stamping and concentrating ...	1·05	$0·38^{13}$	$0·27^{8}$
Cost of freight, smelting, marketing, and New York office, cents. per lb.	$0·34^{9}$	0·33	$0·20^{37}$
Total expenditure per ton of rock treated	8·26½	2·47	1·67
Net profit per ton of rock treated	...	0·22	0·29¼
Loss do.	4·53	...	...

It would appear from this table, that since wages have practically remained unchanged, or are in fact higher (considering what they will purchase), the reduction in the cost of producing copper is due to improvements in mining, in stoping, in drifting, and through the use of rock-drills, high explosives, and other modern improvements, reductions in freight charges, in milling, and concentrating, in smelting and marketing, due to improved systems of doing the work.

All this shows that improved processes, greater facilities for the distribution of products, and greater skill and economy in the management of business have brought about the lower prices of recent years, and with them have added to the general prosperity.

If you look for the cause of these developments of the practical science of mining, it may be traced largely to technical education, the development of technical societies and technical literature. Mr. John Birkinbine, in his presidential address at the Montreal meeting of the American Institution of Mining Engineers,* states that there are to-day in the United States four engineening societies of a national character, with a membership as follows:—

	Founded.	Membership.
Society of Civil Engineers	1865 ...	1,650
Institution of Mining Engineers ...	1871 ...	2,400
Society of Mechanical Engineers ...	1880 ...	1,650
Institution of Electrical Engineers ...	1886 ...	650

* *Trans.*, vol. xxi., page 962.

A few facts selected from many which could be mentioned, illustrate the progress made during the existence of the American Institution of Mining Engineers from 1871 to 1893. The annual output of iron ore has increased from 3,000,000 to over 16,000,000 gross tons. One and two-thirds million gross tons of pig-iron was the output of the blast furnaces of the United States at the birth of the American Institution. Last year shows a total of over 9,000,000 gross tons, while, owing to improved construction and methods, a smaller number of furnaces yield the larger quantity of pig iron; three-quarters of the American product being produced with coke. The growth of the American steel industry is not less remarkable, and the enquiry might be carried further into the manufacture of rails, plate and bar-iron, and steel nails, cars, machinery, and great works fitted with superb appliances for fabricating them. The statement that a ton of pig-iron, of bar or plate-iron, or a keg of nails now sells at from 33 to 40 per cent. of what was received for it in 1871, whilst the price of steel rails is but 25 per cent. of what these commanded in 1871, is an eloquent commentary on the debt mining owes to the development of scientific principles by mining institutions. In 1871, the greatest depth which had been reached in any of the Lake Superior copper mines was 1,000 feet, and the price of copper stood at 30 cents and upwards, yet it was then impracticable to work those mines which did not produce mineral carrying 2 per cent. or more of copper. At the present time there are mines in the same district 4,000 feet deep, and with copper selling for 12 cents per pound, mineral yielding 0·6 per cent. is raised from a depth of 2,000 feet, crushed, jigged, and delivered at refining works, and sold at a moderate profit on the operation.

The hydraulic elevator and the deflector applied to hydraulic mining are inventions of great practical importance, which only date back, the one to 1870, and the other to 1876, and, under favourable conditions, gold-gravel has actually been hydraulicked for as low a sum as 3 cents per cubic yard. Figures will be found in this paper which illustrate the advances that have been made in reducing the costs of quartz mining, and improving the extraction of gold, etc., by chlorination and concentration.

Pan-amalgamation for silver-ores has been bettered and cheapened, and silver-lixiviation, direct matte-smelting, and the cyanide process are all modern processes born of scientific enquiry, which have made important advances and are likely to bear yet better fruit. Silver-lead smelting has been greatly developed, the so-called practical smelter having given place to the chemist and lead-metallurgist, cleaner and better work is now

done than formerly, lower-grade lead-ores are utilized, and lead-slags made in 1878 are now being re-worked.

Advances have been made in the utilization of electricity, the production of aluminium, and a host of other directions for which the world is indebted to the efforts of members of societies like those named in the United States, the various branches of the Federated Institution of Mining Engineers, which numbers over 2,000 members, the Iron and Steel Institute with 1,500 members, the Institution of Civil Engineers with 6,000 members, the Institution of Mechanical Engineers, of about the same size, and similar institutions in Germany, France, and other parts of the world working for a common object.

Any union for mutual advancement commands esteem, so long as the better element of membership is not hidden or overruled by selfish purposes controlling the administration of affairs to the disadvantage of its mutual progressive features. A past president of the Iron and Steel Institnte, alluding to the visit of that Society to the United States in 1890, expresses, the writer thinks, the mission and achievments of technical associations for the advancement of engineering, and of mining and metallurgical knowledge, in words which may be appropriately quoted:— "The expeditions, through which we meet eye to eye and voice to voice our friendly competitors, to discuss the interests and the scientific aspects of the industry which absorbs us, has been of great personal and national benefit. It is thus we learn how much has been accomplished by persistent and intelligent labour, how much remains to be achieved, and how, by the free exchange of ideas, and of productions, friendly understanding is promoted, and personal acquaintance is built up."

The author's fifth proposition (the influence of quality and price of supplies, etc., used upon the cost of milling) needs no demonstration, but in regard to the sixth and seventh (the power employed and the situation of the works), the Disraeli and New Queen mills will once more serve as illustrations, the former showing an economy both in cost of fuel and transport. (Table II.) This may be partly explained by the Disraeli boilers being of tubular pattern, and being run up to their full capacity, avoiding waste of surplus steam,* and as regards

* At the New Queen and other Charters Towers batteries, Cornish and Lancashire boilers are almost exclusively used, owing to the character of the feed-water which incrusts the tubes; and at the former mill (as is often the case) the engine-power is largely in excess of the immediate requirements, being intended to provide for any enlargement of the plant.

transport, the Rishton works enjoyed the advantage of a tramway between the mine and mill. Of course where water-power is available it offers a large saving over steam. Taking the total actual net difference in the cost of milling at the two mills, viz: 3s. per ton attributable to the natural advantages of location, and other causes that have been pointed out, if the one mill has actually cost let us suppose £3,000 more than the other to erect, it is certain that on a steady output of 40 tons per diem, it would have paid this sum back with interest in less than two years by effecting a saving of £1,480 to £1,780 per annum, or about £125 to £150 per month.

There is a good deal also to be said about the eighth factor in the case (the efficiency of the general management), as it is the master-key to the whole business.

If the workman is to demand and obtain high wages in the future it will only be by employing high-class technical and business skill in the management and conduct of operations, so as to take advantage of every practical scientific improvement which tends to cheapen the aggregate cost of production, whilst duly proportioning wages in all departments to relative efficiency and usefulness, whether in respect of mental attainments or manual dexterity. Unskilled labour can no more hope to compete with skilled workmen living in comfortable circumstances (though they be highly paid), in a field where skill is required, such as mining, than the skilled workman can expect to banish cheap labour from a field where nothing but slight manual effort and intelligence are needed.

We have seen that the difference between the cost of working in one mill as compared with another depends on its design, location, quantity and kind of ore put through it in a given time, the number and character of the external appliances for catching gold or other metal after it leaves the crushing machinery, the cost of fuel, labour, and supplies, the power available, and the organization and *personnel* of the staff, which directly depends on the general management. In regard to this last essential the gross cost of operations is nothing to gauge it by, the only certain proof of efficiency and economy is the ratio which the cost of handling and treating a given tonnage, bears to the percentage of gold or other metal saved when compared with other parallel cases.

Technical skill, combined with business management, in dealing with material and with men is absolutely necessary nowadays for the regulation of work, as the general efficiency of the staff as well as the expenditure on supplies, etc., depends upon it. Discrimination in employing the right man and the proper material in the right place, is the pivot upon

which management turns, though it amounts in the former case simply to a knowledge of the time when to put on an extra man at a profit and when to knock one off the pay-sheet, where to employ a mechanic and where to make shift with a labourer—a knowledge only to be gained by experience of men and materials.

Where you have to deal, as you must in any large mining undertaking, with the three different factors of brain, manual skill, and muscle, it is not less ridiculous for instance to place a man, who may be perhaps an adept at spalling stones, in charge of a mill at the salary of a first-class foreman, than it would be to put the latter to cob ore at the wage of a labourer.

Engineering knowledge and business capacity are co-essentials in selecting as well as in managing modern metallurgical processes and mining enterprises. The capitalist who fails to recognize this "knocks a nail from the inside into his own coffin." The miner who does the same, throws a stone at the only bird which can lay "the high-priced golden egg," where, as in some localities, the sum of his daily wage amounts perhaps to a half-guinea or more.

No one can doubt that the question of mutual and fair adjustment between the two co-important trade factors of capital and labour so as to enable both to exist in harmony, is fast becoming the burning problem of the hour, as it certainly will be of the next decade, or at any rate the next century, and it is certain that the prop upon which the very existence of the miner and the capitalist mutually depends is science practically applied. The miner who thinks otherwise is simply footing his stulls on treacherous ground, which will sooner or later break away with him, and entomb himself and his mates in the *débris*, whilst, as for the guinea-fowl who selects the board-room merely for a roosting place, he will equally promote his own extinction, becoming with the decay of all mining enterprise worth the name like "the dodo."

Messrs. McDermott and Duffield remark*:—"It is not possible to supply the lack of skill and experience by instructions and the owners of mills should take the only safe course in this part of the business by employing a good mill-man who has a record of successful work elsewhere. The owners of a mine or directors of a company may be impressed by the bearing and talk of an applicant for the position, but the frequent and lamentable failures resulting from this method of choosing a mill-man are proof of its inadequacy. Testimonials as to character and experience need just as much examination as the applicant

* *Gold Amalgamation*, page 13.

himself. Past successful record in modern mills is the only testimonial of real value. The salary to be paid a good man should be the last consideration. A cheap man is often a ruinous investment, where the final success of a mining investment rests on the successful treatment of the ore."

Another most important factor of success in running a mine in "loyalty." Loyalty to his immediate chief the miner or mill-hand owes to his shift-boss; the shift-boss to the mine-captain or mill-foreman the same as they owe it to the manager or superintendent, and he in his turn is bound to protect his employers' interests, as represented by his directors. As miners, in whatever walk of life we are placed, whether as employers or employed, we owe loyalty to our industry and to one another; both to those dependent on us and to those we are dependent on. Nothing is more fatal to the proper handling of large bodies of men, whether soldiers, miners, or other operatives, than the lack of *espirit de corps* as well as discipline, and a most unfortunate state of things is often brought about by the exercise of mistaken subordinate judgement in opposition to superior instructions definitely laid down. Rules may be technically right or wrong, but a departure from them is only justified under most special circumstances, which were obviously not provided for when the order was given, but individual foresight, and independent action is of course to be commended, and moreover to be expected in the absence of definite instructions.

There is, perhaps, no business which requires more careful management than custom-milling, for the management is constantly confronted with little mathematical problems like these:—If 20 stamps can be run with 12 men, how many men will be required to keep 25 heads on the drop? It is unfortunate you cannot deal with men in fractions except on paper, but it is sometimes possible to solve the problem with a boy. To take another example:—If it costs you 1s. 3d. to cart 1 ton 3 miles on a fine day on a good road, how much will it cost to bring in stone 4 miles on a bush road on a rainy day? To work this out you can reckon that your carts will sink from 6 inches to 18 inches in black soil or mud, as the case may be, a factor of no small consequence to the result.

Very frequent mistakes in deciding upon proper plant and processes have scattered over the world reduction-works of all kinds of which nothing remains but the ruins, though such failures happily are rarer year by year, as more skill and experience is demanded by mine owners in the selection and operation of plants.

The selection of a process, as well as the conduct of it, should be

entrusted to a trained engineer, instead of (as has been too frequently done) leaving it to an amateur; as well put a miner to pick the best fleece out of a flock of sheep!

In the discussion on Mr. Curtis' paper* one speaker remarked on the impossibility of attaching credit to the judgment of managers, like some he instanced who do not consider it worth while enquiring what they are saving or what they are losing, and totally neglect the question of cost; where the ore is shovelled into the battery with a happy disregard of whether it is pounding itself at times to pieces or not. The author heartily coincides with him, but he hopes this will be remedied by more engineers turning their attention to mining; men who can deal equally with the practical, commercial, and technical scientific difficulties of the business, having been alike through the mining school or college, the mine and the mill, beginning of course as assistants or articled pupils,† as it is a grave mistake to suppose that mining can be learnt without a large amount of practical experience of it, both underground and overground, as well as in the office and the shops.

On the other hand, the author has the highest respect and regard for the self-enlightened, self-educated, practical man, than which latter term none in the mining vocabulary is more widely abused or misunderstood. The author knows, personally, men in Australia and elsewhere whom he has the greatest esteem for, who have never had the advantages of a school training, but who were born, not made, engineers, who, by the force of their natural industry, clear perception, and talents, have deservedly raised themselves to the positions in the profession which they occupy and adorn, and he would take the opinion of such men on special points that they have made the study of a lifetime in preference to any other.

They are men, however, who have their heads screwed on the right way, and are of a totally different calibre from the self-assertive, self-advertized, and self-seeking practical man, to whom he has before alluded, who cannot see beyond his natural horizon and scoffs at progress, because in nine cases out of ten he has been brought up neither as a miner nor as an engineer, but has taken to digging late in life, when all other occupations failed.

The author has said enough to show that what applies to labour and management applies equally to machinery and plant, if efficiency be sacrificed to cost; indeed, he should not wonder if the man who framed the

* *Proc. Inst. Civil Engineers*, vol. cviii., page 135.

† The author refers to the system on which colliery engineers are trained, by articling them to a firm of mining engineers in a colliery district.

proverb "penny wise and pound foolish" was a miner who had in his mind's eye the crumbling skeleton of some modern mining company or other which had paid dearly for such a policy in course of time.

One very general mistake to be avoided in milling, as it has ruined more mining enterprises than any other, perhaps, is the erection of a mill, before there is a mine on the spot, sufficiently proved to warrant such a step, *i.e.*, to keep the proposed works employed. Under such circumstances the possession of a mill is a positive disadvantage, as it means, in most instances, that a certain number of men must remain more or less idle part of the time, unless custom ore can be got to keep the requisite permanent staff occupied; under such circumstances it is generally far cheaper to have the ore treated at public crushing mills, if such exist in the neighbourhood.

The writer does not mean, of course, to dispute that the possession of a mill by a private company which can keep it partly, though perhaps not fully at work, does not tend to cheapen production, but the point is that sufficient stone must be supplied by the mine, or from others in the neighbourhood, to keep a certain minimum number of heads constantly employed, and the larger the number, other things being equal, the greater should be the profit. To illustrate the fact that the possession of a mill, if it can be kept moving, assists in cheapening costs the writer would draw attention to a statement made by Mr. L. W. Marsland.* He says (pages 14 and 15): "The point at which quartz ceases to be payable varies, of course, with the circumstances. Thus, in the colony of Victoria at the present time, quartz yielding 10 dwts. of gold to the ton is found to be highly payable, the lodes being of great size. At Charters Towers, the minimum varies considerably. In the great Day Dawn mines 15 dwts. quartz will pay,† as the ore is comparatively free, and the lode (that is to say, the payable portion of it) is very large. As a rule, however, it is understood that, except in the Day Dawn mines nothing under 1 ounce stone will pay; and in some localities such as the Caledonia, it requires a yield of 2 or 3 ozs. to give payable results, the lode being usually very small, and the rock in the walls, which must be taken away to facilitate working the levels, very hard. With the introduction of improved appliances and methods and the consequent reduction

* *The Charters Towers Gold Mines.* Waterlow Bros. and Layton, 1892.

† According to the Report of the Day Dawn P. C. Company for 1891, the cost of mining and milling (excluding general expenses at Charters Towers, £1,348 1s.) came to £1 14s. 3d., equivalent to about 10 dwts. per ton. Including local and London expenses, the author estimates that the company's total costs did not exceed the equivalent of 11 dwts. $18\frac{6}{47}$ grains on an output of 27,416 tons in 1891.

of the cost of mining and milling, it is hoped that the minimum payable return will some day be reduced to such a point as will encourage the working of the many hundreds of reefs on the Charters Towers gold-field, which at present remain unworked, because of their supposed non-payable character."

This statement reminds one of the early days of California.

Dr. Egleston, page 568, remarks: "Prior to the year 1865, it was considered essential that vein matter should yield at least £4 3s. 4d. per ton to be treated with profit, but the introduction of modern machinery has shown that an ore yielding £1 13s. 4d. to £2 1s. 8d. can be mined at great depths and profitably milled when steam is used; and when high-pressure water can be had £1 0s. 10d. is profitable.* Miners' wages are taken at 12s. 6d. to 16s. 8d. per day (10 hours' shifts). Wood for steam costing from 8s. 4d. to £2 10s. per cord, and water-power from 8d. to 1s. 4d.† per ton of ore treated."

A rough division of the gold-bearing quartz-mines of California into three classes may be rationally made.

First, small veins yielding high grade ore in comparatively small quantities whose cost of extraction and milling is £5 4s. 2d. per ton, and the average yield, say about £10 8s. 4d., as an example we may cite the New River district in Trinity County. The quantity of this ore treated amounts to about 5 per cent. of the ore extracted in the state.

The second class would embrace all the large mines, including 75 per cent. of all the ore extracted in the state. The average cost of mining and milling ore of this class, including general expenses, would be about 12s. 9½d. per ton to mine and mill; the Plymouth and the Zeile, on the mother-lode in Amador County, whose ore only assays £1 5s. 9d. to £1 11s. 7½d. per ton, are instances in point. The ore is free milling, carrying 1½ to 2½ per cent. of sulphides which are often very rich, assaying as high as £20 17s. 8d. to £41 13s. 4d. per ton. Shaft-sinking costs £3 2s. 6d. to £3 15s. per foot. Sawn timber £4 3s. 4d. per 1,000 B.M. Round poles 16 feet long, 14 inches to 16 inches in diameter, 12s. 6d. each. Miners wages are 10s. 5d. per day.

The third class, which comprises 20 per cent. of all the ore mined in the state, would include the Bodie, whose grade is rather high, yielding £6 5s. per ton, but expensive to mine and mill on account of the high

* One might even say 14s.

† This is paid to ditch companies, which bring the water long distances.

prices of labour, fuel, and lumber, the average being about £2 10s. per ton. Most of the gold and silver mines of Ingo County may be included in this group.*

We see from this that the cost of mining as well as milling is largely contingent on quantity, but it is equally to be borne in mind that the quantity milled depends more on nature than on man, who, whilst he can control the amount of development (which is a very important factor in the question) is powerless to alter the size and distribution of the ore-shoots in a vein. Mr. Marsland adds to the remarks that have been quoted "Owing to these circumstances the yearly returns of gold from the field, large as they are, cannot be expected to compare with those of other places in other parts of the world, where, in the great majority of cases, the first consideration is quantity of gold without respect to the question of profit! If the same amount of money were expended in the erection of batteries and developing the mines of Charters Towers as has been within the past few years expended on the South African goldfields there would be no difficulty in producing gold at the rate of 100,000 ounces per month."

Now, although Mr. Marsland's statements no doubt apply to the cost in general of local company-mining on the premier goldfield of Australia, as it is sometimes called, other companies as well as the Day Dawn, can mine and mill below the average Mr. Marsland gives, but mostly the English companies, which possess batteries of their own. The Bonnie Dundee and Mosman for example (both under excellent management) should be able to give a good account of themselves, but in proof of this assertion the author will again take the New Queen and Disraeli for illustrations.

Both these companies possessed mills of their own when the author was in charge of them, and their evidence is the more valuable as regards cost of mining, because they represent two opposite cases in the same locality.

The one (the Disraeli) affording an example of a very wide lode formation, 15 to 40 feet wide, carrying irregular bunches of ore, thrown

* In the Transvaal, some of the large mines on the Randt make 10 dwts. pay. At Barberton, the cost of Sheba gold is said to be £1 4s. per oz. obtained (the ore averaging 1 oz. 12 dwts. 22 grains.) In Mysore (where fuel is costly, and the climate is liable to disorganize the native labour at certain seasons when cholera is prevalent) the cost is said to be about £1 18s. 10d. per oz. of gold extracted (the ore averaging 1½ to over 2 ozs. per ton). The actual mine charges at Nundydroog however were less than this, in 1891 amounting to £2 14s. 8½d. per ton.

sometimes to the hanging, sometimes to the foot wall,* which next to an extremely narrow lode, jambed in between hard walls, is certainly the most expensive and variable class of ground to work. The solid feet of ground that must be stoped to yield a certain tonnage being of course greatly in excess of the ore it would represent were the mineralized portion of the vein regular and compact, which all entails extra cost in winning as well as exploratory and dead-work.

The other case, the New Queen cross-reef, is a fair example of a Charters Towers vein, a well-defined quartz lode, averaging from 6 inches to 3 feet in thickness,† with well-defined walls of syenite.

Of course the actual cost of stoping in any two instances, apart from the size and nature of the lode, length of ore-shoot, quality of labour, general management, and character of explosives used, will depend on the system on which the mine is laid out, the ground is removed, ventilated, drained and supported, and the relative perfection of the haulage and other mechanical arrangements for saving labour.

The cost of development and exploratory work, is also liable to still wider fluctuations, and nothing affects the cost of mining more than the proportion which this bears to the producive work (depending on the nature of the lode, and scale of operations), as it obviously stands in inverse proportion to the tonnage of ore that a given area of ground will yield, and varies with the mining experience and judgment of the management; if the factor of luck in exploitation is excluded from the result. More money can be saved or wasted in fact by good or bad judgment in regard to the probabilities of ground, and the method of proving it, than in any other way; the matter rests solely on the mining knowledge of the superintendent and mine captain, based on a sound or unsound acquaintance with the practical problems of applied geology, as bearing upon ore-deposits of different kinds. A knowledge of those branches of geology which deal chiefly with the igneous crystalline and metamorphic rocks, the distributions and occurrence of ore deposits under different conditions; the phenomena of structure and of faults as they variously affect veins is of the utmost benefit and importance to the metal-miner.‡

The cost of exploration and development§ in the case of the New

* Report of Mr. C. P. Purinton, page 5.

† Manager's Report, August 27th, 1888.

‡ In the exploration of mineral ground, the lack of detailed knowledge such as this is likely to prevent the best advantage being taken of natural conditions in laying out the work, and providing for the most economical extraction of the ore.

§ The distinction between the two terms is determined by the shaft, level, or winze being in ore-bearing (payable) or barren (unpayable) ground.

Queen cannot be given as it has not been published, but we can take it at what ought to be a liberal allowance at Charters Towers under similar circumstances to those above stated.

Assuming that each foot of level and winze driven develops or explores an amount of ground equal to 1 foot in length multiplied by half the depth or length of the block on each side of the drivage, multiplied by the average width of the lode between the next adjacent levels and winzes, or the points where such would ultimately be driven or sunk; at the ruling rates of sinking and driving in the district one may reckon that the cost of development and exploration in a reef of the kind described ought not to exceed 7s. 6d. to 15s. per ton on an output of 270 to 500 tons a month.

The cost of mining 1,086 tons (on an average of 270 tons stoped per month) is given in the author's report, dated January 31st, 1889, published by the New Queen company, as £1 8s. 2d. per ton, to which must be added a charge for general expenses, which may be placed* at, say 2s. 5¼d. to 5s. 1¾d., reckoning 270 to 500 tons mined and milled per month.

With regard to the Disraeli mine, taking six months' working, during which 525 tons of ore were won monthly, the cost of stoping may be reckoned approximately at £1 11s. 5d. per ton, but the cost fell to 18s. 6d. per ton, breaking 750 tons a month, during two months in 1887. The underground work was under the immediate charge of Capt. Thos. Blaney, a most competent miner, and all round man.

The cost of dead-work and exploration for the same periods and outputs may be taken at 17s. 2d. to 9s. 3d. per ton, to which we must add, say 6s. 3d. to 3s. 4d. for management and general expenses, which should be a liberal allowance.

The cost in the two cases might be therefore calculated relatively as follows:—

	Disraeli. Ore Mined per Month. Tons. 525			Tons. 750		
	£	s.	d.	£	s.	d.
Development and exploitation	†0	17	2	†0	9	3
Mining (stoping)	†1	11	5	†0	18	6
Milling 890 tons per month	‡0	9	2	‡0	9	2
General expenses	§0	6	3	§0	3	4
Total cost (estimated)	£3	4	0	£2	0	3
	Dwts.		Grs.	Dwts.		Grs.
Equivalent in dwts. to the ton	18		$17\frac{3}{4}$	11		$18\frac{30}{47}$

* Company's Annual Statement of Accounts to June 30th, 1890 and 1891.

† See pages 68 and 69. ‡ See Table II. § Proportions assumed.

	New Queen. Ore Mined per Month. Tons. 270				Tons. 500		
	£	s.	d.		£	s.	d.
Development and exploitation	*0	15	0	...	‡0	7	6
Mining (stoping)	*1	8	2	...	‡1	0	0
Milling 852 tons per month	†0	12	2	...	†0	12	2
General expenses	*0	5	1¾	...	*0	2	5¼
Total cost (estimated)	£3	0	5¾		£2	2	1¼
	Dwts.	Grs.			Dwts.	Grs.	
Equivalent in dwts. to the ton	17	16$\frac{16}{41}$			12	7$\frac{25}{41}$	

Reckoning a pennyweight of Charters Towers gold, as being worth 3s. 5d. on the average, it will be noticed that in the cases given the cost does not come up to Mr. Marsland's general assumed average, and it would be very considerably less on a larger tonnage mined and milled.

It would be a step in the right direction towards making mining more of a business and less of a speculation, if mining companies more often charged and set aside a fixed sum annually for depreciation of the mine, as well as for redemption of capital spent on plant.

Some of the South African companies pursue this sound and proper business policy on the basis that the ore in sight must redeem the cost of its development, for instance, supposing 135,081 tons standing in sight against £24,814 14s. 3d. spent on development for 12 months, a charge of 3s. 8d. per ton would be debited against the stone milled in the year. This is a good policy since mines may, and often do, in fact, become poorer in depth, which may happen in two ways, either by a diminution in the value of the ore, or a reduction in the size of the ore-bodies.

The mine owner is generally anxious to hurry forward the erection of a mill too precipitately, because he knows it is the requisite first step towards making profit at all, unless he can ship his ore, or treat it elsewhere; but what is the result? In some cases it is found, after a large capital outlay has been thrown away, that no mine, worth speaking of as such, exists, and the glowing reports made by some so-called expert, on the faith of which the money was subscribed, turn out worthless. In others, the money which should have been invested first in mining exploration and development is expended on surface improvements, owing frequently to insufficient preliminary investigation and consideration, and the consequent failure to provide adequate working capital. An insufficient balance consequently being left to open-up what might have turned out a paying property, with the result, that the shareholders, tired of calls without returns, refuse to subscribe more money.

* See pages 68 and 69. † See Table II. ‡ Proportions assumed.

In many such cases, if the funds raised at first had been expended on the mine, the profit from the sale of the ore would have sufficed to erect the plant afterwards, and the investment would have been saved from failure, or at any rate there would be the satisfaction of knowing that the mine had been thoroughly tested as far as circumstances permitted and found unable to pay, and in such a case a useless waste of capital would be avoided.

It is perhaps a hard matter for an engineer who has to prepare preliminary estimates to avoid running one of two risks. Either he is liable to underestimate, in which case the company finds itself, as already mentioned, without sufficient funds to complete what it has undertaken to do; or he must allow a certain margin at the risk of being considered extravagant. But the latter course is really the safer risk of the two, and it is the best guarantee, in point of fact, that the interests of his employer is his first consideration.

Mr. H. D. Hoskold* remarks, "It will always be found as a rule that to err on the side of excess of size of machinery . . is far better than defect." Alluding to natural or artificial means of draining a mine, he also adds: "The allowance to be made must depend upon the requirements of the case, and the judgment and capabilities of the engineer in charge of the execution of the works, but it is not unfrequently the case that the hands of a good man are completely tied by the control exercised by a board of directors, who, perhaps for the first time, may have engaged in mining. Such interference is most absurd, and occasionally proves very ruinous to the shareholders, because a really good and efficient man could not work under such restrictions." What applies to the matter in question is relevant to many other branches of mining, where capital outlay is involved.

"A few of the failures† were the result of speculations for purposes not strictly honest, but they have all brought mining more or less into discredit as a haphazard investment, when in reality with the same foresight, prudence, and management which is given to other commercial enterprises, mining and milling would pay more than double the legitimate profits of ordinary business." Three things only are necessary, the co-operation of the capitalist, the miner, and the engineer for one common object—to advance legitimate mining. United, their power to do so may be compared with a treble-force detonator, separate them, and their useful effect is reduced to the power of an ordinary cap.

* *Engineer's Valuing Assistant*, page 10. † Dr. Egleston, *Silver*, page 445.

Speaking generally, the proper process to adopt is that which all things considered, will yield the largest possible commercial profit, a point it is the business of the mining engineer to determine; but one thing is perfectly certain, the process not to adopt, is that which leaves any doubt whatsoever, whether there will be any commercial profit at all.

Profit and economy in the conduct of business on a large scale, such as most large mining corporations demand at the present day, requires capital, combined with business tact and organisation. The secret of the success of American mining ventures, lies in six facts :—

(1) Americans treat metal mining as a business, that is to say, they purchase mining property first-hand on an investment basis, not as a mere share-gamble, a condition of things brought about by the efforts of mining men and the mining press, a tendency we have commenced to follow in England only in recent years to any marked extent.

(2) They are not afraid of risking a large capital outlay to obtain a a big return.

(3) They carefully estimate the chances before embarking in a new proposition, and if they lose by their own want of judgment, they do not blame their manager or engineer, unless he is the responsible person, but pocket their experience and try again with hopes and prospects of better luck, aided by better knowledge. The difference in fact between the company promoter and the capitalist, whether English or American, who interests himself in *bona fide* mining, is that one does justice to his property and his employees; the other does not care a red-cent either for the industry or the men engaged in it, so long as he can "boom his stock," a very proper object, no doubt, providing the shares are worth it.

(4) They are open to adopt any improvement demonstrated to be such, no matter what the first cost, so long as it tends to cheapen production in the long run.

(5) They minimise their risks of loss by developing promising prospects, rather than financing worked out mines, and multiply their chances of profit as much as possible, *i.e.*, avoid trusting all their chickens to one hen.

(6) They will bid and give a high figure for the highest technical and manual skill obtainable in any special province where it is needed, and it therefore pays to employ such; a principle which people in other parts of the world with post-dated ideas and less foresight would call extravagance, but which, with good business judgment, is in point of fact, the very essence of economy.

Table IV. summarizes various particulars already given, in connexion with which the author has to acknowledge the kindness and trouble taken by a number of his friends and members of the profession in aiding him with every information in their power, but owing to the difficulty of collecting particulars with regard to actual gross cost of different classes and sizes of plant, cost per ton of treatment by different processes, and losses in treatment, he regrets that the scope of this table cannot be still further enlarged.

The writer will be thankful to receive, and will gratefully acknowledge, any information or criticisms which will enable him to extend, revise, or perfect its individual details.

He ventures to say that at the present time it is impossible for any private individual to get complete particulars of the kind, in twenty or even ten instances that would illustrate each process, if the whole world were prospected for data.

It is to the interest of all mining-undertakings that such information should be forthcoming, because anything that tends to a knowledge of these matters is a step towards cheapening mining generally, to the advantage of each individual company engaged in its pursuit, and the industry as a whole.

It is not too much for shareholders in mining companies to expect an annual statement of the working costs per ton, subdivided under such general heads, as have been mentioned. If such particulars are not obtainable it is usually, let us hope, from unconsciousness of the importance commercially, to the stockholders, or otherwise one is forced to the conclusion that the oversight is due to defective management or business direction, or else that owing to other causes the affairs of a company will not bear the search-light of open investigation.

In the Colonies and America, much is done by the various governments in the way of collecting statistics of working; but without attempting in any way to pry into the private concern of individual profit and loss, there is still room for doing more, as is evident from the paucity of information on some of the points that the author has touched upon.

Statistics can be made complete without being unduly voluminous, and their collection is well worth the attention of English mining companies and engineers engaged in metalliferous mining abroad.

To make them of practical value it is necessary that, however, they should be collected on the same system. The following table is mostly a summary of facts previously stated :—

TABLE IV.

Process.	General Nature of the Ore Treated.	Daily Capacity of the Works.	Approximate Average Cost of Treatment per Ton. Lowest.	Lowest.	Highest.	Metal or Mineral Saved.	Saving Effected.* Lowest.	Highest.
		Tons.	£ s. d.	£ s. d.	£ s. d.		Per Cent.	Per Cent.
Concentration	Heavy pyritic ores	70—800	E. 0 0 10 A. 0 1 1¾	0 11 10 0 12 6	Blende & galena. copper... ...	60	99
Smelting	Lead, argentiferous and frequently auriferous? ...	24—96	A. 0 12 6	6 5 0	Silver and gold...	90‖	98
Chlorination—Vat	Pyritic concentrates... ...	3½—9	A. 0 14 9½	4 3 4	Lead ... ... Gold ... ... Silver	85 90 60	95 94 65
Barrel	Pyritic concentrates and occasionally "crude ore" ...	2—4 50—200	A. 0 11 10½	3 0 10½	Gold ... ... Silver	92 60	97 65
Cyanide process	Pyritic gold concentrates, tailings, and crude ore	50—250	A. & S. 0 6 10½	2 10 0	Gold ... ... Silver	65 70	84·5 80
Raw grinding	Pyritic gold concentrates §... Pyritic gold ore	2¾—3 20	S. 1 8 5½ E. 0 8 10	2 16 11 ?	Gold and silver	40 78·7	80 91·7
Washoe pan process ...	Pyritic silver tailings carrying gold	150	A. 0 10 3¾	1 6 0	Gold ... ... Silver	25 75·8	42·5 85·8
Pan-amalgamation: Roasting ...	Complex milling ores of silver	20—240	A. 1 13 4	3 2 6	Gold ... ... Silver	40 65·3	60 94
Pan-amalgamation: Dry	Free-milling silver ores	30—100	A. 1 5 0	2 11 0	Silver	70	85
Pan-amalgamation: Wet... ...	Free-milling silver ores		A. 0 12 6	1 17 6	Gold ... ... Silver	45 60	60 80
Boss process	Combined gold and silver ores	120	A. 0 12 6	2 1 8	Silver and gold...	75	85

A.—America. E.—Europe. F.—Africa. I.—India. M.—Mexico. N.—South America. S.—Australia. W.—Wales. * Careless or ignorant management may of course entail larger losses than those given. † Stamps. ‡ Huntington Mill. § With a small alloyage of silver. ‖ Under most unfavourable circumstances.

TABLE IV.—*Continued.*

Process.	General Nature of the Ore Treated.	Daily Capacity of the Works.	Approximate Average Cost of Treatment per Ton. Lowest.	Approximate Average Cost of Treatment per Ton. Highest.	Metal or Mineral Saved.	Saving Effected. Lowest.	Saving Effected. Highest.
		Tons.	£ s. d.	£ s. d.		Per Cent.	Per Cent.
Plate amalgamation—							
(1) With concentration and grinding ...	Ores containing free-gold and pyrites	20—160	S. & I. 0 3 $0\frac{3}{4}$	0 14 6	Gold	45	87·6
(2) With concentration and chlorination or smelting of the concentrates	Ores containing free-gold and pyrites	80—130	A. { 0 1 $5\frac{1}{2}$ / 0 3 3	?	Gold ... ... / Silver	92 / 70	93·8 / 74
(3) Without concentration	Free-gold ores	70—700	W.A.& F.†1/$4\frac{1}{2}$, ‡1/$0\frac{1}{2}$	0 15 $7\frac{1}{4}$	Gold	70	94
Lixiviation— Russel process	Complex silver ores free from *much* lead or copper	75	A. 1 6 3	2 10 4	Silver, etc. ...	82	91·9
„ raw leaching ...	Tailings (raw or roasted) ...	25—75	A. 0 6 $10\frac{1}{2}$	0 19 4	Silver	55	60
Ordinary lixiviation ...	Complex silver ores	50—100	M. 0 15 5 / A. 1 0 6	? / ?	Silver	67	85
Patio process	Rich silver ores, containing traces only of lead and zinc	10—19	M. 5 4 2	5 14 11	Silver ... ... / Gold	70 / 60	90 / ...
Tina process (modified)	Complex silver ores	8	N. 1 8 $7\frac{1}{2}$	...	Silver	...	60
Modified Tina & Fondo (Bolivian)	Docile silver ores ... ... / Base silver ores	8 / 8	N. / N. 1 17 6	...	Silver ... ... / Silver	92 / 80	96 / 85
Pyritic smelting ...	Complex silver and gold pyritic ores	100	A. 0 12 6	?	Silver and gold...	?	96

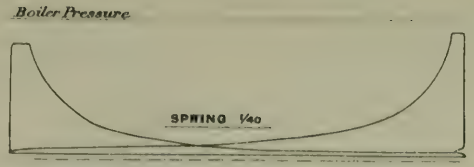
A.—America. E.—Europe. F.—Africa. I.—India. M.—Mexico. N.—South America. S.—Australia. W.—Wales. * Careless or ignorant management may of course entail larger losses than those given. † Stamps. ‡ Huntington Mill. § With a small alloyage of silver. ‖ Under most unfavourable circumstances.

(*To be continued.*)

Federated Institution of Mining Engineers.
Transactions 1896-97.

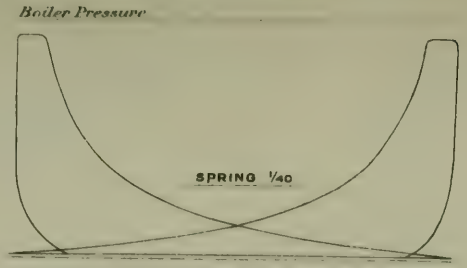
To illustrate Mr. C. C. Leach's Paper on "A Corliss-engined Fan at Seghill Colliery."

FIG. 1.



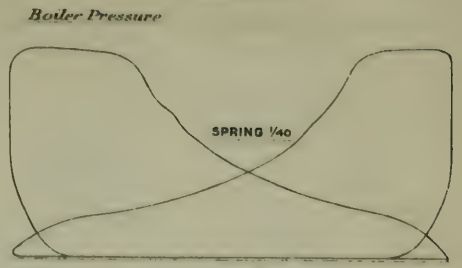
ENGINE RUNNING AT 20 REVOLUTIONS PER MINUTE, AND MINE UNDER ORDINARY CONDITIONS.

FIG. 2.



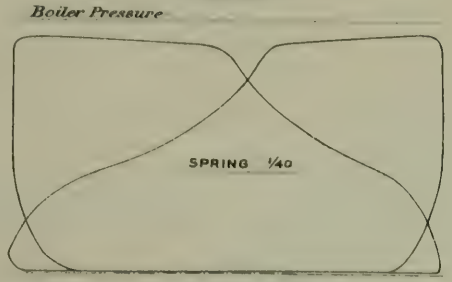
ENGINE RUNNING AT 40 REVOLUTIONS PER MINUTE, AND MINE UNDER ORDINARY CONDITIONS.

FIG. 3.



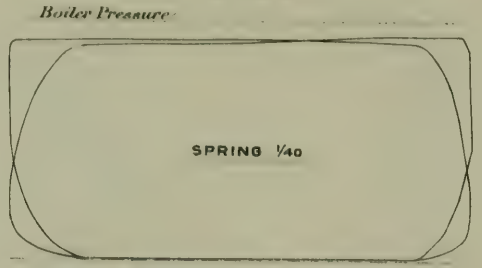
ENGINE RUNNING AT 60 REVOLUTIONS PER MINUTE, AND MINE UNDER ORDINARY CONDITIONS.

FIG. 4.



ENGINE RUNNING AT 60 REVOLUTIONS PER MINUTE, AND SEPARATION DOORS OPEN.

FIG. 5.



ENGINE RUNNING AT 67 REVOLUTIONS PER MINUTE, AND SEPARATION DOORS OPEN.

FIG. 6.

ENGINE RUNNING AT 35 REVOLUTIONS PER MINUTE, WITH FRONT STEAM-VALVE DISCONNECTED AND STEAM ADMITTED TO THE BACK END OF CYLINDER ONLY.

THESE INDICATOR DIAGRAMS ARE HALF-SIZE.

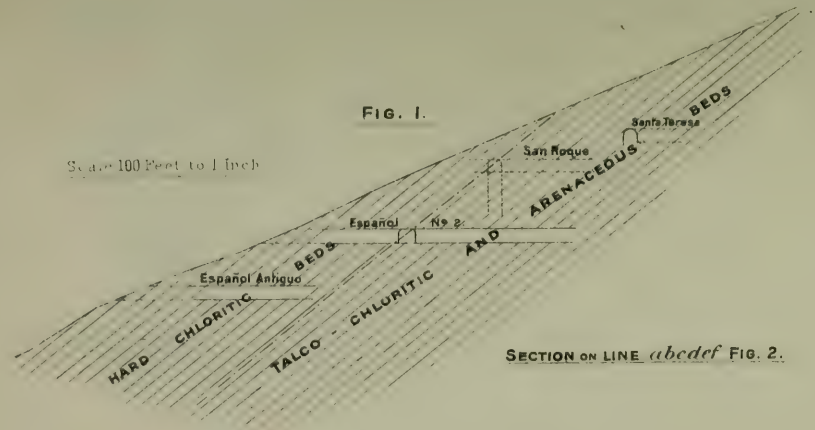
North of England Institute of Mining and Mechanical Engineers.
Transactions 1893-94.

And Mr. Reed, Successor to Lithos, Newcastle.

Federated Institution of Mining Engineers
Transactions, 1892-93

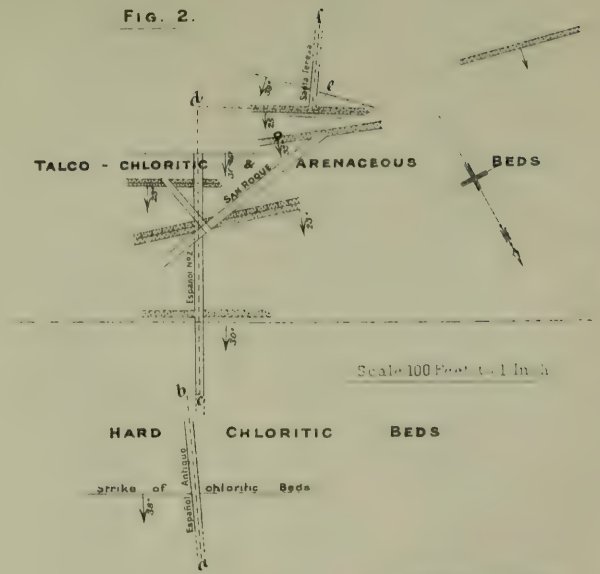
To illustrate Mr Edward Halse's "Note on the Occurrence of Mercury at Quindiú, U.S. Colombia".

FIG. 1.



SECTION ON LINE *abcdef* FIG. 2.

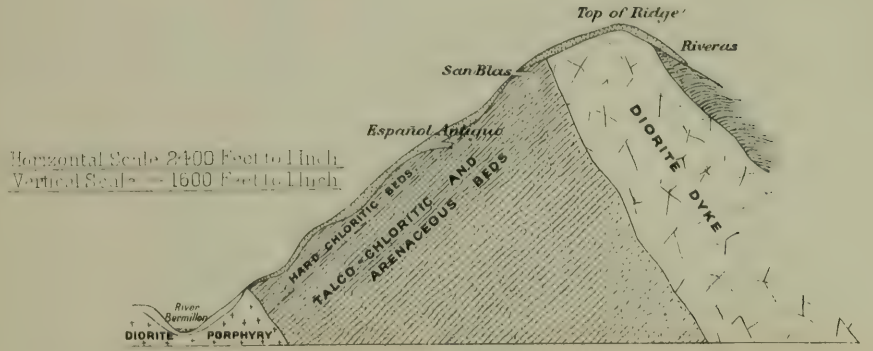
FIG. 2.



REFERENCE TO FIG. 2.

Cinnabar Impregnated Veins or Beds

FIG. 3.



North of England Institute of Mining & Mechanical Engineers
Transactions 1893-94

And.w Reid Sons & Co.e Lithos. Newcastle

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
OCTOBER 14TH, 1893.

MR. A. L. STEAVENSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings on September 30th and that day. He also reported the proceedings of the Council of the Federated Institution of Mining Engineers.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. JOHN WATSON COOK, Colliery Manager Greenfield House, Crook.

Mr. LEWIS EVANS, Mining Engineer, Coetzeestroom Estate and Gold-mining Co., Limited, Kaapsche Hoop, North Dekaap, Transvaal.

Mr. FRANK SIMPSON FERENS, Mechanical and Electrical Engineer, 13, Railway Arches, Westgate Road, Newcastle-upon-Tyne.

Mr. HENRY J. GIFFORD, Superintendent of the Ouro Preto Mines of Brazil, Limited, Mines de Passagem, Ouro Preto, Brazil.

Mr. FREDERICK HALL, Mines Manager, Harrington Road, Workington.

Mr. JOHN HEDLEY, Mining Engineer, Coatham, Redcar.

Mr. JOHN CARY BAKER HENDY, Mining Engineer, Etherley, by Darlington.

Mr. FRANK T. HOWES, Assistant Colliery Manager, Singareni Collieries, Hyderabad, Deccan, India.

Mr. WILLIAM HUMBLE, Government Inspector of Collieries, Coal-fields Office, Newcastle, New South Wales.

Mr. WILLIAM HAMILTON MERRITT, Mining Engineer and Metallurgist, 485, Huron Street, Toronto, Ontario, Canada.

Capt. G. NICHOLLS, H.M. Commissioner of Mines for Natal and Zululand, Pietermaritzburg, Natal.

Mr. RALPH NIXON, Lecturer on Mining to the Northumberland County Council, 47, Holly Avenue, Newcastle-upon-Tyne.

Mr. JOHN SHOTTON, Assistant Engineer, Locomotive Department, Ottoman Railway Co., Smyrna, Asia Minor.

Mr. JAMES JOSEPH TONKIN, Manager of Silver-lead Smelting Works, Linares, Provincia de Jaen, Spain.

ASSOCIATE MEMBERS—

Mr. HERMAN ALEXANDER KROHN, Secretary to the Roburite Explosives Co. Limited, 103, Cannon Street, London, E.C.

Mr. GEORGE FREDERICK MANSELL, Public Accountant, and Fellow of the Institute of Secretaries, St. Mary's Chambers, Church Street, Colchester, Essex.

ASSOCIATES—

Mr. JOHN THOMAS BOLAM, Overman, The View, Beamish, Chester-le-Street.

Mr. JOHN FAIRS, Under Manager, Escomb Bridge, Bishop Auckland.

STUDENT—

Mr. ALGERNON NOBLE, Mining Apprentice, Broomhill Colliery, Acklington.

The following gentlemen were nominated for election :—

MEMBERS—

Mr. FRANK J. AGABEG, General Manager, Apcar & Co.'s Collieries, Sitarampore, India.

Mr. J. A. CHALMERS, Mining Engineer, P.O. Box 357, Johannesburg, Transvaal.

Mr. J. CHATER, Mining Engineer, Bengal Nagpur Coal Co., Limited, Evelyn Lodge, Asansol, Bengal, India.

Mr. FRANCISCO M. COGHLAN, Engineer, Catorce S.L.P., Mexico.

Mr. F. LAWRENCE CORK, General Manager, Bengal Nagpur Coal Co., Limited, Barakur, Bengal, India.

Mr. THOMAS CROUDACE, Colliery Manager, West House, Haltwhistle.

Mr. JOHN ETHERINGTON, Consulting and Mechanical Engineer, 39A, King William Street, London, E.C.

Mr. CHARLES FERGIE, Mining Engineer, Drummond Colliery, Westville, Nova Scotia.

Mr. JAMES FLETCHER, Colliery Manager, Wickham and Bullock Island Collieries, Carrington, New South Wales.

Mr. EDWARD GLEDHILL, Engineer, Carolina Hacienda, Honda, Republic of Colombia, South America.

Mr. WILLIAM GRIFFITH, Manager, The De Beer's Prospecting Syndicates, Waterloo House, Aberystwyth, South Wales, and Fort Salisbury, Mashonaland, South Africa.

Mr. JOHN ERNEST HARDMAN, Mining and Consulting Engineer, P.O. Box 520, Halifax, Nova Scotia.

Mr. ROBERT GEORGE HIGBY, Mining Engineer, Manager, Borrea Coal Co., Limited, Sitarampore, Bengal, India.

Mr. ROBERT G. LECKIE, Manager, Londonderry Iron Co., Limited, Londonderry, Nova Scotia.

Mr. ALFRED BENJAMIN LINDOP, Mining Engineer, Blackball, *via* Greymouth, New Zealand.

Mr. GEORGE HAMILTON LLOYD, Civil and Mining Engineer, Dolgerddon Rhayader, Radnorshire.

Mr. Carr Waller Pritchett, Jun., Mining Engineer, Apartado 84, Pachuca Estado Hidalgo, Mexico.

Mr. C. Robinson, Engineer, c/o Messrs. Wilmard, Spillhouse, & Co., Cape Town, South Africa.

Mr. Alfred F. Seecombe, Mining Engineer, Albaston, Gunnislake, Cornwall.

Mr. Charles William Sibold, Civil Engineer, Public Works Department, Bengal Civil Service, Mirzapore, N.W.P., India.

Mr. Peregrine Oliver Wilson, Mining Engineer, 30, Finsbury Circus, London, E.C.

Associate Member—

Mr. Henry Davies, Lecturer in Geology and Mining for the County of Glamorgan, Treharris, R.S.O., South Wales.

Associate—

Mr. Ralph B. Dobmand, Under Manager, Waldridge Colliery, Chester-le-Street.

Students—

Mr. Wardle Asquith Swallow, Mining Student, Bushblades House, Lintz Green.

Mr. Thomas James Tomlinson, Student, Harton Colliery, South Shields.

CONFERENCE OF DELEGATES OF CORRESPONDING SOCIETIES OF THE BRITISH ASSOCIATION FOR THE ADVANCEMENT OF SCIENCE, NOTTINGHAM, 1893.

The report of the proceedings of the Corresponding Societies Committee of the British Association was read, and of Prof. J. H. Merivale, the delegate of the Institute, as follows:—

Newcastle-upon-Tyne,
September 25th, 1893.

To the President and Council of the North of England Institute of Mining and Mechanical Engineers.

Dear Sirs,

The delegates from the Corresponding Societies met at Nottingham on the 14th and 19th inst. I attended both meetings.

The Chairman, Dr. J. G. Garson, drew attention to the advantages that are enjoyed by the societies corresponding with the British Association. Among these I may mention one which is not perhaps realized by many members of the Institute, viz., a classified list of all the papers published each year in the *Transactions* of the Corresponding Societies is printed in the Annual Report of the British Association.* This list or index fills no less than 96 pages in the Report for last year, and is

* This list is reprinted annually in the *Transactions of the Federated Institution of Mining Engineers.*

obviously of great value to all students. The *Transactions* of the societies are placed upon the shelves of the library of the British Association, where they can be consulted by the members of the Corresponding Societies.

The Earth Tremors Committee wish to make experiments in a mine, and as this is a subject in which the members of the Institute are interested, it might be advisable to purchase one of Prof. Darwin's pendulum seismographs and set it up below ground at a colliery in this district.

With reference to flameless explosives, I reported that the experiments of the committee of the North of England Institute were progressing, but had not yet reached a stage when a report could be published. Prof. H. B. Dixon asked—though not at the delegates' meeting—if we had tried ammonium nitrate cartridges, and advised our doing so.

The Underground Waters Committee hope to present their final report next year; meanwhile, they will be glad of information, which may be sent to Mr. C. E. De Rance, 28, Jermyn Street, London.

The Committee on Erratic Blocks (Secretary, Mr. P. Kendall, Yorkshire College, Leeds) want information from Durham and Northumberland.

Information, too, is asked upon the denudation of high-lying drainage areas by Mr. W. Watts, Oldham Corporation Water Works, Piethorn, Rochdale.

A morning in the Chemical Section was devoted to the question of explosions in mines, with especial reference to coal-dust. Messrs. Clowes, Dixon, Galloway, Hall, Lupton, Thorpe, myself, and others, took part in the discussion.

Yours faithfully,

JOHN H. MERIVALE.

The PRESIDENT read his inaugural address as follows :—

PRESIDENTIAL ADDRESS.

By Mr. A. L. STEAVENSON.

On taking my seat as President of this Institute for the first time, it recalls my earliest recollections of its meetings, its members, and its general surroundings in the old building, then known as the Neville Hall, which stood back about 50 yards from the front street, a two-storied stone structure, black with the smoke of centuries. It had been built by the Baron of Bywell and Bolbeck, and hence had been called Bolbeck Hall; but when its founder in 1378 was created Earl Marshal, it received the name of Westmoreland Place.

It does not seem clear whether that which I remember was the whole of the original building, or whether the Literary and Philosophical Society, in building their library, had already destroyed a portion of it; but after travelling up and around its newel staircase of stone steps, on the afternoon of one of the meeting days, we reached a long low room, almost filled by a table, around which the members sat on cushioned benches.

And there would generally be found in the chair our first President, Mr. Nicholas Wood; on his right Mr. Tom John Taylor, on his left Mr. Doubleday, our first Secretary, and amongst others present Mr. Matthias Dunn, Mr. Barkas, Mr. Edward Potter, Mr. T. Y. Hall, Mr. J. J. Atkinson, Mr. Wales, Mr. J. Easton, Mr. E. F. Boyd, Mr. William Anderson, Mr. Chas. Carr, Mr. T. E. Forster, Mr. J. Marley, Mr. T. G. Hurst, Mr. James Willis, Mr. R. S. Johnson, Mr. W. Southern, and Mr. James Morrison, who have now all left us; but the active part they took in those early days entitles them to be remembered, and it is pleasant to do so. "I am forgotten as a dead man out of mind" is I fear the lot of most of us, but to-day we will make an exception to the rule.

Now, we were exceedingly fortunate in our first President: in Mr. Doubleday's memoir of him, he says:—"At Killingworth colliery, the young man was thrown into the society of one who exercised a considerable influence over his future life; and whose own arduous and successful career his young companion unquestionably assisted to bring to a fortunate issue. This was George Stephenson—then himself a young man—whose persevering ingenuity had already begun to attract attention, and who about

this time was made directing engineer of Killingworth High Pit. In young Nicholas Wood, Stephenson found exactly the coadjutor he wanted. He found his companion endowed with imperturbable good temper, a docile disposition, great power of application, and perseverance under difficulties hardly inferior to his own. Young Wood became accordingly the confidant of Stephenson; the depositary of his plans and schemes of whatever description; and his assistant in that series of persevering experiments which at last resulted in the modern locomotive, which, but for the early and irrepressible energy of Stephenson and his young companion, might yet possibly have remained a desideratum."*

But not only was Mr. Wood the friend and assistant of Mr. George Stephenson during the development of the locomotive, he was also the author, in 1825, of the first work upon railroads, a work now of the greatest historical interest, and filled from one end to the other with valuable experiments upon all kinds of subjects connected with railways. Take for instance page 197 of the second edition, published in 1830, a copy of which he presented to my father, and which for more than forty years has had a place on my bookshelves. He says—"Impressed with the importance of knowing the precise amount of resistance opposed to the motion of carriages along railroads, and also the resistance by different forms of carriages, Mr. George Stephenson and myself, in October, 1818, commenced a series of experiments upon the Killingworth railroad to ascertain that desideratum."

Then we have a lengthy account of experiments upon horse work—upon the strength and deflection of cast and malleable iron rails, others on the friction of ropes on various engine-planes, others on locomotive work, and in fact a treatise, which at the time it was published, was of national importance.

With Mr. Wood as chairman thirty-seven years ago, my own part in the proceedings of this Institute was limited to that of an attentive listener, but within less than three years I was busy with my first paper on "The Manufacture of Coke,"† and from that time to the present I have endeavoured as far as possible to promote the objects aimed at by its founders.

That those efforts have not altogether been unsuccessful, I am proud to recognize in your selection of me to fill this office.

With a view of avoiding repetition and obtaining a proper grasp of the situation, I have again read over the addresses of all my predecessors

* *Trans. N. E. Inst.*, vol. xv., page 49.

† *Ibid.*, vol. viii., page 109.

We have seen how Mr. Wood and Mr. George Stephenson, in the early days of the railway system, having no data to guide them, set to work to ascertain facts for themselves.

Such work is valuable, not merely in itself, but in the experience it affords, and the training it supplies to the experimentalists. What dissection is to the doctor, the practical test is to the engineer, and what he has proved by his own labours day by day, becomes in time the experience, which assists him at every turn, and invests his opinion with all its value.

To the mining engineer especially, in every department of his work, from the boiler on the surface (which supplies his power) to the face of his mine, every part of his machinery, every apparatus of whatever kind, and every material which he uses, should at some time have been tested, and its value accurately ascertained.

An examination of our *Transactions* will show that they largely consist of records of such work, and in that their chief value consists. But while they supply us with information of what has been done by others, they by no means obviate the necessity of proving all things ourselves.

In this work, in the course of my now somewhat long experience, nothing has struck me more forcibly than the difficulty, but necessity, of getting accustomed to exercise strict accuracy: nothing is more common than to find results obtained by experiment diametrically opposed to each other, and facts alleged to have been ascertained absolutely incompatible with known possibilities. Some one, perhaps, has invented a particular mode of firing boilers; he immediately, without any experience in such work, sets about proving that the use of his method results in a saving of 25 per cent. of fuel. Or he puts down a new pumping arrangement, tests it, and claims to be doing 20 per cent. more work than by any other system.

It ought to have been obvious to the man who tested the boiler fires that no form of firebar would alter the quantity of heat given off by a pound of coal, and that the only possible advantage to be gained was in the saving of manual labour, always assuming that the heated gases have been properly supplied with atmospheric air, and not allowed to pass to the chimney until that heat has been exhausted by contact with the boiler.

In like manner, the boiler itself has been subject to perhaps more careless experimenting than any other apparatus. A very large proportion of results so obtained have been found absolutely useless. The tests of a few hours, which are commonly made with the water marked on the gauge, and the fires judged by appearance at the beginning and end of the time,

have little, if any, value, the duration of the experiment having been far too short: such tests should never extend over less than a week, and a month's test would be much more worthy of credit.

Of ventilating fans in the last thirty years there has been no end, and as each new one appeared it was said to have excelled all previous efforts; until, as we have seen, the work done, according to the statement of the persons who tested them, has exceeded the power applied. And the question which arose was, not as to the accuracy of experiments, but to prove how such a result was possible. Let us discuss this inaccurate work in respect to fans a little further.

A powerful machine is newly erected, and after a few weeks' running the manager of the colliery, who is anxious to realize the benefits he has anticipated, the manufacturer of the machinery, who has his interests also at stake, and probably the patentee of the fan, no less wishful to obtain a good result, meet on a certain day to go through the ordinary operations involved in such an experiment. The speed of the fan and the engine must be taken, the water-gauge has to be observed, and the air measured. In all these apparently simple details, and without the slightest intention of dishonesty, it is no uncommon thing to get results 20 per cent. more in favour of the machine than it really yields.

The fan is probably 40 or 45 feet in diameter; and an error to the extent of one single revolution per minute would reduce, to a very appreciable extent, the true horse-power exerted by the engine. The water-gauge may have been placed in a position where it will show 10 or 15 per cent. greater vacuum than it is really putting upon the pit, and the reading of it accurately is also a source of much error; in fact, three persons looking at it and reading it at the same time will very frequently, if they make a note of their observations without communicating to each other what they have got, find their readings differ by as much as one-tenth of an inch. But it is in the accurate measurement of the air where the most fallacious results have been got. The influence of currents at the top and bottom of the drifts, and the irregular flow of air where there are any bends near the point of measurement, make the work one of much more difficulty than is commonly believed. That any man who can hold an anemometer and read it is capable of measuring an air-current without wide experience or thorough instruction is a fallacy of the most serious character. Only in a drift which is fairly straight and of a sufficient length to allow of the point of measurement being at least 20 feet from any cross drift or sharp bend, and divided by strings into equal areas, and by allowing the anemometer to remain in each space, say 30 seconds, can accurate results be obtained.

Now, when we consider all these delicate operations in the hands of such observers as I have already alluded to, and the interest which all conducting the trial have in obtaining a good result, it is easy to understand how these errors are propagated.

But when once the fact that the work is not simple, but demands the strictest care has been grasped, one half of the difficulty has been overcome; and it is in the training and experience which such experimental work affords that a large share of the amount of advantage consists which is gained by the engineer.

It is, perhaps, not necessary to follow this subject further; but the same lesson which is inculcated by the testing of a fan is taught in whatever department he is engaged.

It is obvious that these remarks naturally lead us to consider how the engineer can be best fitted to perform these duties, and, in fact, how he should be educated.

We have seen within the last twenty-five years a College of Physical Science founded in this city, which will, I trust, go far to remedy the rule of thumb which has prevailed so much in the performance of work such as has been described. In speaking of this valuable institution, I wish, at the earliest moment, to express my heartiest recognition of the good work done personally by its professors in the assistance they have at all times given to this kindred society. I cannot make any distinction amongst them, although I am sure I speak nothing more than the universal feeling of the mining and mechanical engineers when I express my very great regret that we are about to lose from amongst us Professor Garnett.

But the college alone will not suffice, and I will, therefore, venture to describe what the experience of more than forty years has led me to consider as the most suitable training.

Assuming that a youth has been well educated, and reached the age of sixteen years, he will be able to read and write his own language correctly, he will have obtained the rudiments of French and German, and as much Latin as will enable him to read Virgil or Cicero, together with mathematics, so far as, say, simple algebra. At this time the sooner he gets a couple of years of thoroughly good pit experience the better, and nothing will obviate the necessity of a master to show him what work he ought to do, and to see that he does it.

No college will ever supply the experience so obtained. Moreover, the position he will hold in life involves such questions as the value of work and the cost of production, he must have acquaintance with such matters as the measurement of work, of the letting of bargains, of pit bills, and

the management of men, surveying and the keeping of plans, and the practical questions which arise in the ventilation of the pit. None of these things can be gained at college, but after two years of pit work he should have a couple of years there, and he will realize the advantage which his pit work has given him in the knowledge of what the information is he requires ; he will be able to understand the value of the subjects he is studying, and having in this way reached the age of twenty, another good year of pit work will find him at twenty-one thoroughly accomplished, and wanting that experience which time can alone supply.

He will, however, have first to pass the examination and obtain his certificate, and if what is called his leisure time has been well spent, this he will easily do. Having myself, since 1872, when the Act requiring these examinations was first passed, acted as one of the examiners for the South Durham, Cleveland, and Westmoreland districts, I feel entitled to say a few words on the subject.

The greatest benefit which these examinations have conferred has been of an indirect character, for, whereas previously a young man gained the practice afforded by his pit work, only in a few cases was it accompanied by that theoretical knowledge which is to be obtained from books. The pit shift being over in the morning, with perhaps a survey plotted in the afternoon, the newspaper or the pipe filled up his remaining time ; but with an examination to face, such subjects as geology, chemistry, and mechanics have to be thought about and studied, and when once the rudiments are acquired, in many cases a taste is awakened for information about which he otherwise would have known nothing. That which began in the nature of a school lesson becomes an absorbing recreation for perhaps a lifetime.

Let it, however, not be supposed that, in speaking as I did a short time since of rule of thumb, I deprecate in any degree self-taught knowledge, as the following will illustrate. Thirty years ago we had no college to assist us, and after ten years in the colliery district, where I had done perhaps more than the ordinary share of surveying which falls to the lot of a young man, I took charge of some mines in Cleveland. There, with new plans to make, it was soon evident that the ordinary miner's circumferentor would not suffice ; I could not read the theodolite, and had not the least knowledge of the bookkeeping or calculations it involved. But having procured an instrument, without accepting the assistance of any of the works of instruction upon the subject, and being able only to calculate the sine and cosine of an angle, I made out a method of doing the work, so that in two or three weeks I had made a survey of

more than 4 miles from start to finish and over some of the roughest ground in Cleveland, and I was able to tell by calculation that there was an error of 7 links in the survey.

I might, perhaps, have been taught at a college in lessons of a few days that which took me as many weeks to discover for myself, but the satisfaction and pleasure experienced as each feature of the investigation revealed itself well repaid the labour spent in personally acquiring the information.

Before leaving this subject, I will say a few words upon the technical education of Continental engineers as compared with our own. It has been very often said that we were being left behind by our brethren on the Continent. Now, I have had many opportunities of meeting with French and Belgian engineers under circumstances which permitted me to form a fairly just opinion of their qualifications. I have read their mining literature, translated and published some of their works, and I have certainly failed to find them, when taking them all-round, as we say, superior to ourselves in their capacity for the ordinary, everyday work of colliery management.

I am inclined to say that, whilst the English colliery viewer of forty years ago had good practical skill alone, and the Continental engineer to-day excels somewhat in theoretical attainments, we have here that happy blending of science and practice which renders the Englishman, although by no means perfect, the best man of the two.

I will now briefly refer to the work which lies before us. We have seen during late years powerful trades unions established in all classes of labour. I am by no means prepared to say that their influence has been upon the whole injurious; for they have to a considerable extent introduced order where there was before chaos. Arbitrations, joint committees, and the leaders of the workmen in this district at all events give us something tangible to work with, but nevertheless as these organizations become more powerful and less labour can be bought for the same wage, it devolves upon the engineer to invent such mechanical aids as will enable him to reduce the amount of manual labour he requires, and work his mine without increased cost.

Still more important is the work which this Institute was especially founded to promote, viz., that of the safety of workmen. The inspector of mines for the Durham district has more than once brought under our notice that falls from roof and sides produce more than one half of the accidents. In his report for the year 1890, after quoting the special rules upon the subject, he says, "Yet in face of these stringent regulations, no

less than 56·75 per cent. of the whole of the deaths in the district took place from falls during the past year, many of them I regret to say from sheer carelessness and disobedience to orders."

They are a class of accidents very difficult to prevent. I have seldom made an underground examination during which I have not had to caution workmen that they were then incurring danger, and insist upon precautions being taken; all that can be done is to caution and educate the workmen, and punish all who break the rules which are instituted for their protection.

The class of accident which ranks next in importance are those known as "miscellaneous underground," and which in 1891 were 24 per cent. of the whole, those in shafts 12 per cent., on the surface 10 per cent., and for explosions 5¼ per cent.

Upon the manager rests the responsibility of leaving no means untried for improving his own experience, and perhaps in no way can this object be better attained than by a constant attendance at the meetings of this Institute, where every novelty and improvement in the art of mining is brought before us and discussed.

We may, however, point to the statistics of accidents which have occurred since the founding of this Institute with much satisfaction, for we find that, at that time, the ratio of persons employed to each death was 233 to 1, whereas in the last four years, which include 1891, the deaths averaged 580 to 1, or just 2½ times the number of persons employed per fatal accident, an improvement for which we may fairly lay claim to a considerable portion of the credit; the averages for Durham and Northumberland being 895 persons per life lost.

No subject has been more constantly brought before us of late years than that of the value of electricity for lighting and transmission of power, and after what may be termed a youth of wild extravagance it is now, I hope, settling down to a maturity of reasonable and quiet usefulness. Its value and its drawbacks are being proved and admitted.

So far as its lighting value is concerned, it is doubtful whether it has much use for the mining engineer; as the quantity of light he requires for surface operations is comparatively small. But if what may be termed fire risks can be efficiently overcome, it will prove valuable where a good light is required in the screening and cleaning of coals. Many attempts have been made to produce a portable miner's lamp. For my own part I am so thoroughly satisfied with the safety of the lamps we now use, that I have not taken much interest in the attempts to solve the electrical difficulty. Before leaving the subject I should like to emphasize what I

have said upon this question on former occasions, and which I know has been often misunderstood. It was to the exaggerated and utterly misleading statements as to the efficiency of work done by electricity, which were for a long time promulgated, that I objected, such statements being based solely upon the the difference of power put into the dynamo by the belt, and measured in electrical efficiency at the motor.

Now, the cases are very, very few in which it is possible to utilize that power in the motor directly. In, say, a pump intermediate gearing must be used on account of the speed of the motor, causing much loss, and hence results of experiments most recently brought before us have entirely confirmed the views which I had expressed, and which must be obvious to any impartial observer, but more particularly was I annoyed by these electrical efficiencies being placed in comparison with results got where water actually lifted had been compared with the work done by the steam in the cylinder. I am able to quote if necessary scores of these cases in which electricity is said to give 65 to 75 per cent., whilst compressed air gave 30 per cent. But as has been well said, "Magna est veritas et prevalebit." For the transmission of power over distances exceeding 1,000 yards it has proved itself a valuable helpmate, although the first cost of electrical plant will, I hope, be capable of reduction; whether it is a suitable medium to take into a mine beyond the lamp-station or caution-board those who adopt it must decide for themselves.

The coal-dust question has agitated the minds of all connected with collieries in the last few years, and there are some I believe who still shake their heads and refuse to admit the possibility of any of our great explosions having originated through dust alone. The matter seems so clear to me that I have taken very little interest in the later discussions on the subject. There is and can be no dispute of very serious explosions having occurred repeatedly where no gas could possibly have been present. I will mention a few cases just to confirm my statement. In 1886, an explosion occurred in flour mills at Leith, when three persons were killed and four injured. In 1882, an explosion occurred in a flour mill at Macclesfield; a large part of the mill was levelled with the ground, and damage to the extent of £5,000 was done, the engineman being killed. In 1887, an explosion occurred in the United States in some wood-working establishments at a point where the dust and shavings were gathered together: the roof of the shaving-house was blown off and two of the workmen killed. In 1885, an explosion took place in a rag mill at Drighlington, causing the death of two workmen. This was an explosion of the dust at what was described as "the shaker."

There was no possible gas present in these cases, and when we remember that explosion is only a rapid form of combustion to which the fineness of the particles of dust make it peculiarly susceptible, and that coal-dust if heated contains 10,000 cubic feet of gas to the ton, it is difficult to conceive how anyone can still harbour a doubt on the subject. The case seems analogous to that of those who, for many years, whenever a boiler explosion took place, insisted that there was something about boilers we did not yet understand, and it was often asserted that electricity had something to do with it. To such we can only extend our sympathy.

We have lived at a period when events have marched rapidly. In our Vice-President (Mr. Wm. Armstrong) we have the champion of the battle of iron *versus* hemp ropes: a strike of thirteen weeks was required at the Wingate Grange colliery in 1843 to convince the workmen that iron ropes were safe, and at its conclusion the coalowners, fitters, and agents gave him a dinner at Durham in recognition of his victory. I trust he will some day give us an account of the state of coal-mining in his early days, when candles, 60 to a lb., held the place of the safety-lamp of to-day, when coal was brought to bank in corves, and haystack boilers supplied the steam. Such reminiscences are not merely interesting, but often prove useful. There was a case in the law-courts a short time ago in which the question of when the mode of working out all the coal instead of leaving small pillars was first practised, and no precise evidence could be found. So we never know how incidents, which are now common knowledge, may some day prove important, when all record of them has been lost. Let us seek to make the *Transactions* a storehouse to which future generations can always refer, and which will serve to hand down our names and the work we have done to those who come after us.

In conclusion, I would urge strongly upon all, whether owners or engineers, the duty of assisting this Institute in every way; let the former recognize that in the efficiency of his agent (which it promotes) his deepest interests are involved, and the latter see that the profession he has adopted is raised and honoured by the world at large, in proportion as it takes its place amongst societies distinguished for the science and learning of their members.

Mr. G. B. Forster said he was sure the members would all join with him in according to their new President a most hearty vote of thanks for his inaugural address. Mr. Steavenson had commenced with their early years and their first President, and taken them through the various

points which had attracted his notice as being for the good of the coal trade and for the scientific advantage of the Institute. Young men in particular would profit considerably by laying to heart what the President had said about accuracy of observation and deduction. They must join with the President in the hope that the Institute would continue to prosper and be of use to the members generally.

Mr. M. W. PARRINGTON hoped he might be allowed, as the President's senior pupil, to second the vote of thanks. He could bear out what had been said as to the value of education for mining students. When he went to serve his time with Mr. Steavenson twenty-nine years ago he had not the advantages which were now afforded by Colleges of Science, although he must at the same time admit there was not the same competition. Competition no doubt was a very healthy thing, and would lead to greater efforts on the part of mining students at the present day.

Mr. EMERSON BAINBRIDGE said that, as an old friend of the President, he had come that day to have the pleasure of listening to his address, with which he was sure they must all be gratified. It was well known that of all their members no one had been more an exponent of accuracy than Mr. Steavenson himself, and he might venture to define the President's chief characteristic to be a determination to prove the accuracy of his conclusions, although this resolve might sometimes result in the destruction of some pet theories, causing him to change his views. He thought on the question of general education, that it was desirable to develop more and more instruction in mechanical engineering rather than to overdo the pit-work. If out of a five years' tuition two years were devoted chiefly to pit-work and three years devoted to mechanical engineering, he thought such a division of time would tend to make better colliery managers than had been produced in the past. This was shown in the training of the French mining engineer, who, though he might not have an equal knowledge in the practical management of a colliery, could lay out the complete plans of a large colliery plant, being better able to deal with questions of mechanical construction than was generally the case in England.

The vote of thanks was carried with acclamation.

The PRESIDENT, in acknowledging the vote of thanks, said that he had had considerable difficulty in making up his mind to accept the position of president on account of the deafness with which they all knew he was afflicted, and which made it difficult to hold such an office. Relying, however, on the kind assistance of his friends he could not forego the honour, and he would be glad if at the end of his term he should find that the Institute had not suffered by his holding the office.

DISCUSSION ON MR. V. C. DOUBLEDAY'S PAPER ON "THE SUSSMANN ELECTRIC LAMP."

Mr. V. C. Doubleday exhibited and described the Sussmann Miner's Electric lamp.*

The President said the Sussmann lamp appeared to be a very good form of electric lamp, but it had the defect of being without an attachment for indicating the presence of gas.

Mr. J. L. Hedley (H.M. Inspector of Mines) said they all knew that under certain conditions and so long as the incandescent filament was excluded from the atmosphere an electric lamp was safe, and no doubt if the use of such lamps were considered advisable they would look out for the best one. The point raised by the President was a very important one, and for his own part he thought a great risk would be incurred by introducing such lamps unless provided with some means of ascertaining the state of the atmosphere in the mine.

Mr. Doubleday said that the lamp as now constructed did not indicate the presence of gas.

Mr. W. O. Wood said he had had some experience with electric lamps, and the rincipal difficulty he thought connected with their use was with regard to the charging. He presumed that a dynamo would be required for the purpose of charging and recharging the lamps. There was also a difficulty in ascertaining whether a secondary battery was properly charged or not ; sometimes a lamp had been issued partially charged, with awkward consequences to the user. He thought that the glass-cylinder of the electric lamp would probably be of the same strength as any of the glass lamps ordinarily in use, and even if the outer glass were broken there was still the protection of the glow-lamp globe inside. The use of a means of indicating the presence of gas was perhaps hardly necessary for general use by the ordinary workmen, inasmuch as there were so many examinations now made by deputies and other officials who could be supplied with an apparatus to ascertain whether the places were safe or not. In the case of a sudden outburst of gas there would still be little risk, but the deputy would discover it on his return visit.

Mr. J. L. Hedley could only partially agree with Mr. Wood's remarks. If the deputy happened to be present when the outburst of gas occurred he would, of course, warn the workmen ; but taking into consideration the long intervals that frequently occurred between his visits, he thought it important that all lamps should indicate the presence of gas (either

* *Trans. Fed. Inst.*, vol. vi., page 264.

fire-damp or stythe), and show the workmen the state of the atmosphere. He did not want to say a word against the electric light, for he thought it was an improvement, in many respects, on the type of lamp now in use, but it would be a great mistake, he thought, to use a lamp that might lead to something, perhaps not worse than had more than once occurred, in case of an inrush of gas, but to more serious consequences in what may be called the ordinary working conditions of the mine.

Mr. W. C. Blackett said he was in the habit of carrying an electric lamp, and could corroborate Mr. Hedley's warning. The increased light was a great convenience, but he never felt very safe in going about alone with it. He had to travel in both fiery and stythy mines, and he did not care to go into either unless he was accompanied by a man with an ordinary lamp. He quite agreed that it would be unsafe to leave a man alone with an electric lamp in a stythy mine—less safe than in a fiery mine—for many hours.

Mr. J. C. B. Hendy agreed with the previous speakers as to the inadvisability of adopting such a lamp generally without any means of testing for the presence of gases. Some time ago he had headings going through a large fault and a considerable length of disturbed ground, and although there was a strong air-current they had found much gas previously. He would certainly not have placed electric lamps in the hands of the workmen in those headings unless they had been fitted with some means of showing whether gases were present. He did not think it would be wise (as Mr. Wood suggested) to trust to the deputy's periodical visit with a safety-lamp, as within half an hour or less there might be a sudden outburst of gas with serious consequences. At the same time he thought, apart from gas-testing, that electric lamps were a step in the right direction.

Mr. C. J. Murton suggested that the difficulty might be overcome by attaching to the electric lamp, or fixing into a recess, a miniature oil-lamp of the ordinary shielded gauze type, and with a correspondingly small wick and flame. Such an addition would only add a few ounces to the weight of the lamp, and the presence of either fire-damp or stythe would be made apparent in the usual way.

Mr. G. B. Forster noticed that the weight of the electric lamp was 3 lbs. 11 ozs., or about 7 ounces heavier than the ordinary Marsaut lamp. In the case of the workmen the additional weight of the lamp was not very material, as they did not carry them for a long time. A heavy lamp was, however, a drawback so far as the manager of the mine was concerned. He thought this lamp was better than others in which a lens

was placed in front, and consequently only threw the light on one point, leaving the remainder of the space in darkness. There were two points into which discussion might be divided: first, that of safety; and secondly, the question of an efficient light. With regard to safety, he thought the glass in this lamp was not (as Mr. Wood had said) quite as thick as a Marsaut lamp glass, although in this respect it could be improved; but the greatest objection on the score of safety was no doubt that which had been referred to—the lack of some means of detecting the presence of fire-damp or stythe; and until some such means were provided he did not think it would be expedient to introduce the lamp into their mines, although no doubt the light given was a very good one.

Mr. J. A. Ramsay asked if any observations of pit-gases had been made through a prism with the view of obtaining the qualitative composition of the atmosphere seen through, or if a spectral analysis could not be obtained by an application of prisms to decompose the beam of light from the lamp?

Mr. M. Walton Brown enquired as to the length of time required to recharge the batteries? He understood that a secondary battery required considerably more time to recharge than the lamp would burn when charged; and, as a consequence, in the case of lamps burning more than ten hours per day, a double set of lamps would be required, so that one set could be recharged while the others were in use. There was a difficulty in the application of indicators of the presence of gases, as the electricity used to work a Maurice, Liveing, or Swan fire-damp indicator was much greater than that required to heat the glow-lamp filament, with the result that the battery of any electric lamp would be run down in about half an hour if used continuously for gas-testing. These fire-damp indicators were of no use in testing for stythe, and some other appliance must be used for its discovery.

Mr. Doubleday said that assuming a lamp was used for nine hours in a mine, it could very safely be recharged in eight hours, but having in view the length of life of the battery, the more slowly they were charged the longer the life of the lamp.

The President then proposed a vote of thanks to Mr. Doubleday. They were pleased to see the lamp, which appeared to be an improvement on others which they had seen, but whether it would be more enduring or not, time alone would show.

Mr. G. B. Forster seconded the vote of thanks, which was adopted.

DISCUSSION ON MR. LEACH'S PAPER UPON A "CORLISS-ENGINED FAN AT SEGHILL COLLIERY."*

Mr. J. C. B. HENDY said that the paper was very interesting, and it showed that while attention had been given to manometric efficiencies and volumes, the important matter of the efficiency of the entire plant had been somewhat neglected. In the experiments he (Mr. Hendy) had made at Teversal collieries† the amount of coal and water used was recorded, and he noticed in the discussion which followed the reading of Mr. Leach's paper that the amount of water or steam used at Seghill collieries was compared with the amount used at the Teversal colliery, showing that a saving of over 75 per cent. had been effected at Seghill. There was a great difference between the conditions at the two collieries. He thought that the chief source of saving was due to the feed-water being heated by the exhaust steam. Teversal was an old colliery and it would have been difficult to have applied such an arrangement. At Teversal, the feed-water had a temperature of only 60 degs. Fahr., while at Seghill it was stated to be 115 and 172 degs. It would be interesting if Mr. Leach could state the saving between using water heated to 172 degs. and at the ordinary temperature of say 60 degs. He (Mr. Hendy) thought that the coal used (and not the water entirely) was the measure of the efficiency of the engine and boiler, and it was as important to look to the boiler as to the engine.

Mr. C. C. LEACH pointed out that he purposely omitted from his paper the amount of coal used, and the weight of coal burnt was only mentioned in the discussion. It was not fair to charge the engine with the difference which might result from the use of bad coal as compared with good coal and of one boiler with another. The heat in the steam used at Seghill colliery was 1,300 degs., and as the difference of temperature of the feed-water was only about 90 degs., the consumption of coal would only have been one-fifteenth more, if the feed-water had been used at a temperature of 60 degs. instead of 150 degs. The loss would have been very small, say $\frac{1}{4}$ lb. of coal per indicated horse-power. He might add that the temperature of the feed-water did not affect the weight of steam used per indicated horse-power.

Mr. HENDY said he thought that the Seghill engine was the most economical one used for driving a fan that he had heard of.

Mr. LEACH said the economy of the engine was obtained by getting the full pressure of steam on the piston at all speeds, automatic and

* *Trans. Fed. Inst.*, vol. vi., page 48. † *Ibid.*, vol. iv., page 474.

instantaneous cut-off, and the short length of the ports. It was thought undesirable to erect a compound condensing engine for the power required, more especially as the builders guaranteed an economy which other makers said could only be obtained from a compound condensing engine.

The PRESIDENT said that no further remarks being offered the discussion would now be closed.

DISCUSSION ON MR. EDWARD HALSE'S "NOTE ON THE OCCURRENCE OF MERCURY AT QUINDIÚ, TOLIMA, U.S. COLOMBIA."*

Mr. ARTHUR L. COLLINS (London) wrote that in addition to the useful list of cinnabar deposits, contained in Mr. Halse's paper, that there were two other occurrences which had come under his notice. At Zalathna, in Transylvania, small veins and impregnations of cinnabar have been very extensively worked in sandstone. The richer parts were worked as poor men's mines, leaving large quantities of low-grade stuff which would require more elaborate plant for treatment. At Kilkivan, near Gympie, in Queensland, more or less cinnabar has been found over a very large area, and when he (Mr. Collins) was there in 1890 he found some pounds weight of rolled pebbles of pure cinnabar (from a pea up to a large walnut in size), by panning in a small gully cut by the rains in the subsoil. The country rock is conglomerate, of Carboniferous age, lying nearly horizontal, consisting of pebbles and fragments of quartz porphyry and other volcanic rocks, with a crumbling ferruginous cement. It is pierced in places by dykes of diorite, and by many large vertical fissure veins of dolomite and calcite, often brecciated. The veins (one of which has been worked to depth of 200 feet) are often quite free from metallic minerals; but sometimes they show brown hæmatite, blue and green carbonates of copper, and cinnabar in patches, or as stains in chalcedony and calcite. In one vein a mercurial fahlerz is also found, in quantities sufficient to be used as an ore of mercury. The ore, as raised, is mostly very poor, and would require the erection of a large furnace for its treatment. No vein has as yet been followed down through the conglomerate, which is of unknown thickness.

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* *Trans. Fed. Inst.*, vol. vi., page 59.

DISCUSSION ON MR. E. S. WIGHT'S PAPER ON "QUEENSLAND COAL-MINING; AND THE METHOD ADOPTED TO OVERCOME AN UNDERGROUND FIRE."*

Mr. T. E. Forster said that Mr. Wight's paper was interesting, from the fact that it contained a record of the early days of a coal-field, which may some day be of considerable importance. The Ipswich coals seem to be worked under very similar conditions to those of the Barrum river, which he had visited in 1887. The quality of the Queensland coals is generally considered to be inferior to that of the New South Wales and New Zealand coals, but it was quite possible that further development might bring improved quality. At present the limited market was naturally a great bar to any rapid increase in the output. Underground fires had, he believed, given a good deal of trouble, and were generally caused by the small coal left in the workings. Mr. Wight stated that the natural heat of the mines was great, and it would no doubt be interesting to have some particulars of underground temperatures taken in these collieries. Mr. Wm. Fryar, inspector of mines for the southern division of Queensland, took some observations a few years ago in various gold-mines at Gympie, about 100 miles or so north of Ipswich, and found that, so far as his experiments went, the surface temperature in December was, generally speaking, higher than the underground temperature taken in places where there was practically no air-current passing. The deepest shafts observed were the Inglewood, Golden Crown, and Great Monkland. In the first mine, the surface shade temperature was 90 degs., and that at the 1,200 feet level, 78 degs.; at the Golden Crown, surface, 96 degs., and 1,200 feet level, 80 degs; and Great Monkland, surface, 80 degs., and 1,400 feet level, 76 degs.

The following "Note on the Antimony Deposit of El Altar, Sonora, Mexico," by Mr. Edward Halse, was taken as read:—

* *Trans. Fed. Inst.*, vol. iv., page 548.

NOTE ON THE ANTIMONY DEPOSIT OF EL ALTAR, SONORA, MEXICO.

By EDWARD HALSE.

Some years ago, a heavy yellow mineral, found in abundance in the hills 50 miles west of El Altar, Sonora, was amalgamated for silver, and, as it was found to be very refractory, samples were sent to England for examination and analysis. The unknown mineral proved to be stibiconite, a kind of antimony ochre or hydrated oxide of antimony containing 75 per cent. of metal and 5 per cent. of water of combination. Stibiconite had hitherto only been found as a slight coating on some antimony minerals at certain localities in Bavaria, Arkansas, and Borneo, and its occurrence in workable quantity was therefore of great scientific interest.

About thirteen years ago some capitalists of Boston (Mass.) secured the control of nine antimony claims in the district. Each claim measured 800 by 200 metres—on three of them an outcrop of solid antimony was described as being traceable for many hundred feet in length.

While the workings were yet of a superficial nature they were briefly described by Prof. E. T. Cox, of Tucson, Arizona.*

For a time the deposits were actively worked, and a small cupola furnace was erected for the reduction of the ore at Oakland, San Francisco.

In 1882, a description of the process employed, with some further notes on the nature of the deposit, was written by Mr. J. Douglas, Jun.† Just previous to this, "owing to a disagreement with the mine owners and a consequent stoppage of supply," the works were closed down, and since then the mines appear to have been worked more or less intermittently.

In the Federal Government and State maps, the deposits are generally marked a few miles from the Gulf of California, on the eastern flanks of the Sierra del Alamo Muerto, which skirts the sea-shore.

Mr. Cox says that the geological features are similar to those of southern Arizona, and the mountains run in short narrow ranges, having

* *American Journal of Science and Arts*, series 3, vol. xx., 1880, pages 421-3.

† *Engineering and Mining Journal* (New York), 1882, pages 121-2. A brief description, taken from the Journal, is given in Mr. A. J. Phillips' *Treatise on Ore Deposits*, 1884, pages 566-7, but, by an error, it is placed under the heading of "United States."

for the most part a north-and-south trend. The crests are rugged or well-rounded cones. Between the ranges exists a *mesa* or tableland formed of the *débris* of the mountains. The material is so loose and porous that the small amount of water which falls through it leaves the land dry and arid.

Several species of cactus and mesquite, and small trees of hardwood known as *palo fierro* constitute the monotonous vegetation of the hills and plains.

To the north-west a wide desert region, the Desierto of the maps, separates the mines from the coast.

The mountains are of granite, flanked by sub-carboniferous limestone, in most places so crystalline as to obliterate all traces of fossils. Protruding through these and forming the mountain-peaks are porphyry, quartzite, basalt, diorite, and trachyte. Mr. Cox states that the country rock in the immediate vicinity of the antimony mines is quartzite and limestone—while that of the particular deposits examined by the writer appeared to be an altered eruptive rock (basalt or diorite).

The Boston Company controlled an area from 5 to 6 miles long and $\frac{1}{2}$ mile wide. The veins are described as running nearly north and south and dipping east at a sharp angle, from 4 to 20 feet wide, and down to 30 feet below the surface the fissures were "filled from wall to wall with oxide of antimony, almost pure, and remarkably uniform in character."

In 1882, three systems of veins had been discovered within an area of about 4 square miles, viz.:—

1. San José, the most northerly, and most productive in antimony —non-argentiferous.
2. Santa Margarita group—argentiferous.
3. Argentina group—argentiferous.

A specimen well-coated with chloride of silver yielded 120·75 ounces of silver per ton. The silver lies chiefly as a distinct film (greenish in freshly-broken specimens, but which darkens on exposure to light) between the joints. It can be seen by the eye on compact broken surfaces.

The mines, examined about two years ago by the writer, are about 112 miles (43 Spanish leguas) west-north-west of the Santa Ana depôt of the Sonora railroad, nearly 50 miles west of the town of El Altar, and probably 30 miles (11 leguas) from the port of Las Salinas, and 10 to 15 miles from the coast. The deposits are found on hills about 700 feet above sea-level, and running parallel to the Sierra de las Palomas. The mineral occurs in loose stones (*boleos*) near the surface, and in veins,

of which there are several, running north-and-south, north-east and south-west (generally with a westerly dip), and east-and-west (with a southerly dip). The veins frequently intersect, and at these points the mineral usually occurs in bunches or pockets (*bolsas*).

In two of the mines, a fair amount of development work has been done. In one claim, a shaft has been sunk on the vein to a depth of 84 feet.

Two levels or *frontones* have been driven from this shaft on either side, proving the lode for about 100 feet in length. The vein appears widest and best in level No. 1 north-west, where it is 3½ feet thick, and composed of good mineral. On the opposite side of the shaft, the vein is 3 feet thick; here another vein intersects it, forming a pocket of mineral, and at this point the principal vein runs almost horizontally for about 18 feet, and then resumes the normal dip.

At and near the bottom, the vein is more quartzose, and bears much less mineral than in the upper level.

The general run of the veins is north-east and south-west (magnetic 1891) with a dip to the south-east of 52½ degs.; it has a fairly regular width (2 to 3½ feet), strike, and dip, and at one point has a very good and well-defined hanging-wall (*respalda*), but it varies a good deal in richness from point to point, and consists of bands (*hilos*) of black, yellow, grey, or brown stibiconite frequently much mixed with quartz.

The country appears to be soft and decomposed, and is probably volcanic. It varies from white to dark red in colour.

In an adjoining claim, a vein runs east-and-west, with a sharp southerly dip. A shaft has proved the vein to a depth of 150 feet, and shows it to be composed of bands of dark-coloured stibiconite and quartz. The latter increased in depth; the ore from the bottom of the mine consisted largely of this gangue and was too poor to work.

In another claim, one vein runs north-and-south and dips west, consisting of heavy yellow stibiconite, and another north-east to south-west dipping north-west.

A parcel of 50 tons of hand-picked mineral shipped to England yielded as much as 53 per cent. of metallic antimony, and sold for £10 2s. 6d. per ton.

It is estimated that the cost put on cars at the Santa Ana depôt would be about £6 per ton, which is prohibitive.

If practicable, it would be far cheaper to send the mineral to the port of Las Salinas, Gulf of California, a distance of about 30 miles, and to load on barques bringing merchandize from Europe. The total cost, including freight to Europe, would then be about £6 per ton.

Water is very scarce in this region, while, although there is probably sufficient wood (chiefly *palo fierro*) for the timbering of small shafts, galleries, etc, and for raising steam, or for making charcoal if necessary, large timber would have to be imported from the States. No doubt several thousand tons of this mineral could be raised at and near the surface by opencuts and by sinking, but it would seem from the trials already made that the mineral does not continue in depth, so that a large and constant output is by no means assured. The deposits, in fact, would appear to form the capping of quartz-veins, and it is quite problematical whether, below the water-level, the hydrated oxide will be found to give way to the ordinary sulphide (stibnite).

Mr. James D. Dana* describes stibiconite as "massive, compact, also as a powder and in crusts. Hardness, 4 to 5·5; specific gravity, 5·1 to 5·28.† Lustre, pearly to earthy. Colour, pale yellow to yellowish white, reddish white. Composition, probably $H_2Sb_2O_5$ or $Sb_2O_4 + H_2O$. Analysis, oxygen 19·9, antimony 74·5, and water 5·6, total 100. Usually more or less impure. . . . Pyrognostic character, in the closed tube gives off water, but does not fuse; on charcoal decrepitates, fuses with difficulty to a grey slag, and gives a white coating."

The Sonora mineral breaks with a rough conchoidal fracture. Prof. S. P. Sharples says it is only slightly soluble in hydrochloric or nitric acid, or in aqua regia; that fusion with bisulphate or soda only partially resolves it; that it readily decomposes in a platinum crucible with carbonate of soda. It is, moreover, easily reduced in a crucible with powdered charcoal or cyanide of potassium, giving as a single operation buttons of "star antimony."

According to Mr. Douglas the cupola used for smelting the Sonora ore was a round, close-fronted and water-jacketed furnace of the following dimensions:—36 inches in diameter, 6 feet from the tuyéres to the feed door, and 18 inches from the tuyéres to the bottom of the crucible. The tuyéres had a slight dip, the blast was from a No. 4 Root blower. The charge consisted of 80 lbs. of ore, 100 lbs. of slag, and 30 lbs. of coke. The slag was returned until it became too acid, when 10 lbs. of pure iron-ore was added to the next charge. The metal and slag were tapped into slag-pots. Seven tons of ore was on the average consumed by the furnace in twenty-four hours. An enlargement of the furnace-stack helped to save the antimony. The yield was $77\frac{2}{3}$ per cent. of the assay value of the ore, or 40 per cent. of antimony.

* *System of Mineralogy*, 6th edition, 1892, page 203.

† A specimen from Sonora gave 5·07.

The coarse antimony was refined in a crucible—each being heated in a separate furnace. The charge in each crucible was 40 lbs. of coarse antimony, 2 lbs. of sulphide of antimony (pure native stibnite), and ½ lb. of carbonate of potash. The contents of the crucible were poured out into square moulds of the same size and shape as the English ones. The surface was beautiful and deeply starred. The slag removed from the cake was returned without addition to the refining-crucible, and to ten charges more, with the addition each time of a little stibnite and potash. The cakes were finally cleaned by scrubbing, which was a tedious operation.

In the autumn of 1881 the works were turning out about 10 tons of antimony weekly.

The metal obtained from the Sonora ore was pure, containing, according to Mr. F. Claudet of London, only 0·54 per cent. of impurities.

Another analysis gave impurities as follows:—

	Per Cent.
Arsenic	0·036
Copper	0·052
Lead	0·538
Iron	0·039
Zinc	none
Cobalt	traces
Nickel	none
Sulphur	0·254
Total impurities	0·919

The following paper on "The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment," by Mr. A. G. Charleton, was taken as read:—

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part V.—Gold-milling.

The writer will now examine the differences in details of milling practice which obtain in different parts of the world, considering the local conditions and character of the ore treated, in order to account for the diversities which exist.

General Details and Practices in California.

In this connexion, the writer begs to acknowledge his indebtedness to a paper on the subject by Mr. J. H. Hammond,† in the *Eighth Annual Report of the California State Mining Bureau.*

The California ores are mostly quartzose, carrying free gold and iron pyrites, sometimes accompanied by arsenical and copper pyrites, but more frequently galena and zinc blende. Auriferous tellurides and some of the rarer minerals are occasionally met with, but they are of little economic importance.

The vein-filling is sometimes countrified (consisting of wall-rock more or less altered) but is usually quartz, in some instances associated with calcspar, which, however, but rarely forms exclusively the vein-filling or gangue.

The value of the Californian gold ores varies between 14s. 7d. and £1 13s. 4d. per ton in the low-grade ores, up to between £3 2s. 6d. and £6 5s. per ton in those of high grade ; £2 to £2 10s. is probably a rough average of the grade of ore at present treated. The percentage of sulphides (mostly iron pyrites) runs from 1 to 5 per cent. of the ore milled; 2 per cent. being an average approximation in all probability of its pyritous contents.

The saving of these sulphides, though small in quantity, is an important feature in the treatment of these ores, and therefore the majority of the mills of the State have their plants specially adapted to close-saving in this respect, although the value of the concentrates is subordinate to the free gold present in the ore.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351; vol. v., page 271; and vol. vi., page 69. † *The Milling of Gold Ores in California.*

The average value of the concentrates being high, accounts for this, and may be put at £16 13s. 4d. to £18 15s. per ton. In the low-grade ores the gold occurs disseminated in particles through the ore, rarely visible to the naked eye. In ores of high-grade it is often found massive and sometimes in laminæ along the planes of division of the quartz (ribbon-rock), while in other cases it assumes the form of wire (filiform), and is also occasionally arborescent.

Specimen ore is often sold to jewellers, who pay from £4 3s. 4d. to £5 12s. 6d. per ounce for the free gold that the quartz contains. The pyrites is generally massive, though sometimes found crystallized, but iron pyrites of the latter character rarely carries much gold. The sulphides in the country-rock are likewise of little value.

In some of the Californian mills the rock-breakers are arranged in a way somewhat out of the common, but which is a very excellent one. The ore coming from the mine is discharged in the usual way on to a grizzly which separates it into two classes; what passes through goes to the main ore-bin (underneath at the back of the battery), while the coarser rock is retained in what is called the coarse ore-bin, behind the rock-breaker. The grizzly is set so as to form the bottom of the upper half of this last-named bin, the lower part and sides being boarded in.

A shoot leads from the door of the coarse bin (which is worked by a rack-and-pinion) into the jaws of the rock-breaker. By this arrangement the delivery of the ore is controlled, ensuring an almost continuous supply of stone to the machine, which greatly increases its capacity and reduces expense, as it saves the labour of a feeder which is necessary, where, as in most mills, the ore is discharged over the grizzly on to the rock-breaker floor, and also maintains a regular feed.* The only point to be observed is that the stone is sent to the mill spalled to a size which will all enter the jaws of a given-sized crusher.

At the North Star mill with this arrangement, one (15 inches by 9 inches) breaker crushes 30 to 40 tons of hard rock in 5 to 7 hours, effecting a saving in wages of two or three men as compared with the labour required ordinarily, while evidence of its uninterrupted work is shown by the fact that it requires 12 instead of 8 horse-power as is usually computed. The crushed ore joining the fines which have passed the grizzly, is well mixed, which ensures uniformity in the charges fed to the stamps.

* In some of the newer German concentration-works a reciprocating-table, at the end of the shoot, actuated by an eccentric, is employed to ensure this result.

Where fall permits, it may sometimes be found advantageous to crush coarse in one rock-breaker, and deliver the product to a second one, breaking it finer. This greatly increases the capacity of the stamps. The rock-breakers are usually set to crush to 2 or 3 inches. The shoes and dies last from six to eight months: and when made of steel they run about twice as long.

Self-feeders.—The use of self-feeder machines saves a large amount of labour, increases the capacity of the battery from 15 to 20 per cent., besides effecting a considerable reduction in the wear of screens, shoes, and dies.

Other conditions being equal, low feeding increases the capacity of the stamps, that is to say, the ore should be fed in regularly, and in small quantities at a time. When thrown into the mortar irregularly in large charges, the ore is piled up in the box, and the stamp cushions upon it, which impairs the effectiveness of the blow. The best hand-feeding is superior to mechanical devices, but men are mortal, and either through ignorance or inattention, caused by the extremely trying conditions under which such work must be performed for long spells at a time, the machine on the whole is the most reliable, apart from the monetary question of saving in labour where labour is dear.

Automatic feeders are of various types, those with a revolving-carrier (such as the Hendy Challenge) deserving the preference for wet, clayey and sticky ores. Those with a shaking-table, such as the Tulloch and the Stanford roller-feeder, give satisfaction, however, with certain classes of ore, and being the cheaper, are frequently employed. The Challenge feeder costs £52 1s. 8d. in San Francisco, occupies 23 cubic feet packed for shipment, and weighs 750 lbs.

The Hendy improved Challenge suspended-feeder does away with the underneath framing of the ordinary type of machine, and renders the feed-side of the mortars more accessible, the machine being supported on parallel tracks overhead, between the battery-posts and ore-bins. It is specially adapted for this reason for feeding rolls.

Mortars.—This important feature in battery construction depends on the character of the ore and deserves more consideration than it generally receives. Narrow mortars accelerate the discharge of the pulp, and are therefore disadvantageous, where it is desired to amalgamate on inside-plates, or settle coarse-gold in the box. They also occasion an excessive breakage of screens, adding to the expense of renewing and changing them, if the ore is of a hard flinty nature.

The liability of breakage can be reduced by raising the height of the discharge, but this counteracts the advantage aimed at in using a narrow mortar, viz., large capacity. Each mortar is fitted inside, with side and end-linings of cast or wrought-iron to protect it from wear. These linings when of cast-iron last six to nine months. In some mills they are held in place by bolts with the heads countersunk in a recess in the plate, which are tightened up from the outside; a better method, however, is to halve the plates together at the ends.

In many Australian batteries, a length of about 9 inches is cut out of the middle of the front-side of the mortar, and replaced by a dove-tailed iron casting of the same shape and thickness as the section removed. This slides up and down (bearing against a shoulder) when the screen frame is taken out, and facilitates cleaning the box at a general clean-up. When the sides and bottom of the false-section are properly planed and the opening trued up to correspond, the mortar remains perfectly watertight.

The height of discharge, *i.e.*, the vertical height of the lower edge of the screen above the die, should be regulated to correspond more or less with the width of the mortar, taking into consideration the object aimed at in the treatment of any special ore. Narrow mortars require a relatively higher discharge to avoid breakage of screens, and to prevent scouring the inside copper-plates. Having fixed on the proper height of discharge, it is most important to keep it uniform. There are various ways of doing this, which are referred to elsewhere. With ores quickly crushed and readily discharged, it is advisable to raise the height of the discharge, to retain the pulp in the mortars long enough for its proper amalgamation. Where an ore contains a high percentage of sulphides, and cannot be advantageously amalgamated in the battery, or where stamps are used with a view to large capacity in crushing, and it is desirable at the same time to catch the coarse-gold liberated, by allowing it to settle in the box, or in the third case of an ore containing brittle sulphides, which from being subjected to lengthy stamping are liable to be slimed, double-discharge mortars may be found advantageous. Stamps crush faster than the screens ordinarily discharge the pulp, with the result that many of the particles are subjected to unnecessary pulverization, sliming the sulphides which can with difficulty be settled, producing float-gold, and when coarse occasionally perhaps pounding it; hence double-mortars sometimes serve a necessary purpose. Their objections and limitations are: (1) where copper-plates are used inside, mortars of this kind are inconvenient, and are too roomy for efficient amalgamation; (2) owing to the extra water

they use, they are prejudicial to the outside plate-amalgamation, and in some cases to the subsequent concentration, depending on the character of the concentration-method, and the machines adopted to that end. The feed-opening of a mortar should extend nearly the whole length of the box, and be about 4 inches wide, allowing for the upper part of the back-lining.

Screens.—In California, various kinds of screens are used—steel and brass wire-cloth; slot and needle-punched (tough but soft), planished Russian sheet-iron (which has a smooth glossy surface), and tin. These last are not so thick as Russian iron and consequently admit of more rapid discharge than Russian iron for the same diameter of perforations, but they do not last so long. The numbers by which steel and brass wire screens are designated correspond with the number of meshes per lineal inch. The most common sizes in use in California are 30 and 40, No. 30 being made of No. 31 wire and 40 of No. 33 wire, the latter costing in San Francisco 1s. 6d. per square foot. Each battery takes 3 to 4 square feet.

The screens wear most at their lower edges. Brass wire-screens last 10 to 14 days. High discharge and wide mortars promote the life of the screens. One brass wire-screen, No. 30, bar accidents, will put 120 to 140 tons of ore through, before it is worn out. Steel wire is liable to rust, and is therefore not much in favour.

The area of discharge in wire-screens is much greater than in slot or punched iron, producing a more uniform pulp, hence it is in a condition particularly adapted to after-concentration. The numbers and sizes of needle-punched and slot screens correspond to the number of the corresponding needles which will pass through the openings. Nos. 5, 6, 7, 8, and 9 are the commonest sizes. The slots are horizontal or diagonal (angle slots). They are generally burred on the inside to prevent clogging, as the opening enlarges outwards, but this tends obviously to reduce the discharging capacity of the battery, from the fact that a particle of ore which does not strike the opening straight, has no chance of getting through; that is to say, either obliquely, or when flowing down the grating.

Russian iron screens last 15 to 40 days, a month being about the average. The screen-frames are of wood, and should have a batter of about 10 degs. projecting outwards at the top; they rest in a recess cast in the front of the box, and are held in place by wooden wedges driven under claws cast on the mortar. A strip of blanketing should be laid round the frame.

A piece of heavy canvas is generally hung across the front of the mortar.* Stamping should be carried just far enough for the proper liberation of the gold and sulphides from the veinstone. When the gold is finely divided, as generally happens in low-grade ores, the stamping must be proportionally finer than when the gold is coarse. When the sulphides constitute an important factor in the value of the ore it is desirable to crush coarse to avoid excessive loss by sliming them. That dead-stamping is a serious and by no means visionary fact is attested by the statement that generally over 80 per cent. of the ore discharged through a No. 30 screen will pass a No. 60, and often 50 per cent. will pass a No. 120.

TABLE OF SCREEN SIZES.

No. of Needle.	Corresponding Mesh.	Width of Slot. Inches.	Gauge of Russian Iron. No.	Thickness of Iron. American Gauge. No.	Weight per Square Foot. Lbs.
5	20	0·029	14	$23\frac{1}{4}$	1·15
6	25	0·027	13	24	1·08
7	30	0·024	12	$24\frac{1}{2}$	0·978
8	35	0·022	11	25	0·918
9	40	0·020	10	26	0·827
10	50	0·018	9	27	0·735
11	55	0·016	8	28	0·666
12	60	0·015	8	28	0·666

Russian iron is sold in sheets measuring 28 inches by 56 inches, equal to 10·88 square feet. Some manufacturers' measurements differ somewhat from the table. The Attwood screen-measure is very convenient for determining the size of orifices. For inside battery amalgamation, the mortar is provided with strips of silver-plated copper-plate, back and front, seated on wooden blocks (provided with iron-facings where the keys come outside), resting on top of and flush with the inside liners (an Australian term for the lining-plates). Silver-plated brass screws are used to fix them on.

Drop of the Stamps.—This depends on the character of the ore, the speed and weight of the stamps, and the duty required, varying from 4 to 9 inches; 6 inches being about the mean in California. Sufficient drop must be given to produce a good splash. Soft and highly mineralized ores need a low drop usually. The order of drop is of importance, as on it depends an even distribution of the pulp on the several dies. Adjacent stamps should never drop consecutively, as this occasions accumulations of pulp in the end of the mortar, by which the efficiency of the stamps at

* In Australian batteries, a cover of sheet-iron or wood, hung on hinges (cast on the top front of the box) is used.

that end is reduced, whilst those at the other extremity are liable to pound iron. The order, 1, 4, 2, 5, and 3 gives a good splash, and gives satisfactory results. 1, 5, 2, 4, and 3 is also extensively adopted. The remedies, for a bad order of drop, are either to change it, re-cutting the key seats in the cam shaft (by no means a simple affair), or else to suffer the misfortune of having to give one stamp a greater drop than the others.

Duty of the Stamps.—This term, which is applied to the quantity of stone crushed per head by the battery in 24 hours, will depend on the weight of the stamp, number of drops, height of drop, height of discharge, size of screen-mesh, area of screen-opening, dimensions of the mortar and character of the ore. Clayey and hard ores reduce the duty. $2\frac{1}{4}$ tons per stamp in 24 hours is about the average duty of Californian batteries.

Speed of the Stamps.—Low speed is necessary with heavy stamps and high drops. In California, 900 to 950 lbs. stamps, with 6 to 7 inches drops, are run at 85 to 95 drops per minute. With double cams, the speed must not exceed a certain rate, for if the revolution of the cam does not give sufficient time for the stamp to fall, the descending tappet striking the toe of the returning cam, is very likely to cause a breakage or dislodge either boss, shoe, or tappet. A fast drop produces a good splash in the mortar.

The cam-shaft is generally made of mild steel, and the cams themselves may be made of hard chrome steel. The curve of the face should be a slightly modified involute.* The cam-shaft pulleys are best built up of wood, thoroughly seasoned, and the joints filled with white lead and oil. They are held by 3 feet cast-iron double sleeve-flanges. The hub is bored and fitted to the cam-shaft and fastened with a steel key. The outside flange is bored and fitted to the sleeve with a jib-headed steel key. Kiln-dried sugarpine is a favourite material for making these pulleys. Cams ought to last several years. Tappets should have both faces turned true, and be fixed on with a steel jib and two steel keys, bored accurately with the jibs in place, to fit the stems, and counterbored opposite the jibs by moving the centre $\frac{1}{4}$ inch, taking a cut $\frac{1}{8}$ inch deep, with a diameter $\frac{1}{8}$ inch less than the bore. This gives three bearing-points and allows the tappet to move easily on the stem.

* Split cams in case of breakage are more handy to change than solid ones.

The working-face, with the exception of a small annular ring next the stamp-stem, is subjected to equal wear, hence this central part is recessed out $\frac{3}{8}$ inch to $\frac{1}{2}$ inch wide and deep on both ends, so that the cam may not be worn down by the projecting ridge which would be liable to be formed on the inside of the tappet. Where one face is worn, the tappet is reversed. The tappets, like the cams, are usually made of hard steel.

The revolving cam in lifting the tappet causes it to partially revolve, communicating a certain amount of rotation to the stamp as it drops. Its chief importance is in equalizing the wear of the shoes and dies, which promotes better crushing. With an ordinary amount of grease on the cam, a stamp should make a complete revolution in four to eight lifts, but if there be an excess of grease, or the tappet has been allowed to become grooved, the rotatory motion is much impaired. The average life of the tappets is four to five years. They may be broken by being keyed too tight. When their faces are worn they are planed down. The importance of keeping the height of discharge regular is second only to maintaining a fixed height of drop. A frequent examination should be made to see that the cams have not shifted, which is very liable to happen if the tappets are not properly secured. The cams must be greased sufficiently, but not in excess; nothing is more disgusting or liable to interfere with amalgamation than grease being thrown on to the aprons or falling on the boxes.

The wooden covers of the mortar are no protection against oil, as the writer has seen it in some mills running down the stems; he may say that they were not American mills. Wooden finger-bars, fitted into latch-sockets lined with leather, hung on a wrought-iron jack-shaft at the back of the battery, are mostly used in America to hang up the heads. To perform this operation neatly, requires some skill, the battery-man standing on the platform at the back of the stamps just below the jack-shaft, and inserting a wooden lever with his right hand under the tappet as the cam comes round. This lifts the tappet higher than usual, allowing him to slip the point of the finger-bar underneath the back of the tappet, before the cam drops the stamp.

In Australia, a different method is generally employed. Two flat straps of iron are fastened to the battery-posts horizontally, a little higher than the full height of the throw of the cam (one at the back and one in front of the stamps). Resting across these straps are five rectangular bars of iron attached to each battery, provided with a hook at one end and a handle at the other. The hook slides on the front strap, and

the battery-man, taking the handle at the back in his hand and resting a short crowbar on the back strap, catches the tappet as it is raised, prizes it up slightly, and slips the cross-bar sideways underneath.

Shoes and Dies.—These are either of iron or steel. Ordinary cast-steel used for this purpose has a great tendency to chip and cup; recently, however, chrome-steel has come into the market, and in remote districts, where transport is an important item, it has replaced iron.

Steel shoes with iron dies wear more evenly than if both are steel. The life of steel is about two-and-a-half to three times that of iron shoes and dies, while the cost is about twice as great. The mixture of old chrome-steel shoes and dies with iron, produces shoes and dies which wear considerably longer than iron, and where there is no other means of utilizing the old castings this plan may be advantageously introduced. Old ordinary iron castings are sold to local foundries for 1½ to 2 cents per lb.

The shoes are made to fit into the stamp-head by being tapered so as to fit into the core of the socket, the shank of the shoe being surrounded with a number of dry hard-pine wedges attached to it with a piece of twine, upon which the boss is driven on by allowing the head to drop on the bare die. In Australia, in some mills, in place of wood-strips the shank is covered with canvas or blanket in strips, crossed at right angles, overlaying one another alternately, and tied round with thread. The heads or bosses are accurately bored at the top to receive the tapered end of the stem, and have a conical socket cored out at the lower end for the shank of the shoe. Transverse rectangular key-ways, 1 inch by 2½ inches, cross the centre of the head at right-angles, passing through the bottom of the recesses, in which the stem and shoe-shank fit, for the purpose of wedging these latter out with loosening-keys.

The durability of the shoes and dies is affected by the weight, speed, and height of drop of the stamps, the material they are made of, coarseness of stamping, and height of discharge, as well as the manner of feeding the ore and the hardness of the stone. Iron shoes of good quality will last 30 to 47 days; they are worn down usually to from 1½ inches to 1 inch in thickness, and then weigh about 20 to 50 lbs. The dies being protected by the bed of ore that covers them to a depth of 1½ to 3 inches wear longer, but when the length of the shoe is greater than the height of the die the actual life of the latter may be the shorter of the two.

The consumption of iron for shoes and dies per ton of ore crushed is from 1½ to 3 lbs. To obtain a maximum duty, the dies should be kept as high as is compatible with the safety of the screens, and with successful

battery amalgamation; if such be practised. Uniformity in the level of the dies is important, for should one die project much above the others no pulp will remain on it, and the shoe will consequently pound on the naked die. Stems are made tapered at both ends so as to be reversible, and are usually of mild steel. When broken they can be swedged or planed down, and additional lengths welded on if necessary.

Foundations, etc.—Nothing about a battery is more important than well-constructed solid foundations. The mud-sills and battery-blocks are generally made of yellow-pine, free from sap, or sugar-pine. Sugar-pine or red-spruce is used for the framing of the battery. Sun-cracks are filled with melted sulphur or Stockholm tar, laid on while hot. They should be bedded on concrete or on the clean bed-rock. An overhead travelling-crane on a suspended track should extend the full length of the battery. The battery-blocks for each mortar in California are generally made in two pieces each 30 inches thick, and wide enough to fill the space between the line-sills and battery-posts. They are secured with oak keys 4 inches wide, tapered from 6 inches in thickness at the head to 5 inches at the point as well as with six 1 inch bolts. Yokes of 10 inches by 10 inches timber are fitted and bolted to the line-sills and battery-posts. In loose ground the sides of the pit for the battery-blocks may be walled in at the sides and ends. Once the whole set of blocks is level sighted along the whole line (which is generally effected by nailing a board along the front, a short distance below the top, and sawing the projecting part off), the top is planed smooth, making it about $\frac{1}{16}$ inch hollow to prevent it from becoming rounded. It should be covered with sheet rubber $\frac{1}{4}$ inch thick, or tarred mill-blankets and sheet lead ($\frac{1}{8}$ inch thick) to form a seat for the boxes. Personally, the writer prefers the former. The mortars must be levelled true, both lengthways and across, inside.

Amalgam.—In battery amalgamation, the largest proportion of the amalgam is caught on the inside-plates, representing, however, a greater percentage if the gold be coarse, than when it is finely-divided. Width of mortar, height of discharge, nature of plates and attention given to them, and fluidity of the amalgam, all influence the ratio between what is saved inside and on the outside-plates. Sometimes from these causes more is caught without than inside the battery; generally speaking, however, in California 50 to 80 per cent. comes from the battery. The fluidity of the amalgam is generally caused by overfeeding quicksilver, but in tropical climates it is to some extent influenced by the season; being more fluid in hot weather.

The battery-amalgam is invariably richer than the plate-amalgam, its value increasing with the coarseness of the gold, and as might naturally be supposed decreasing with finely-divided alloyed gold. At the Original Empire and Star mills in Grass Valley, California, while the battery-amalgam averages £1 15s. 5d. per ounce, the plate-amalgam only runs 18s. 9d. Amalgam varies greatly in different clean-ups with ores from the same mine. The frequency of cleaning-up depends on the richness of the ore and local practice. The richer the ore, the more often it is necessary to clean-up.

The outside-plates and sluice-plates are usually cleaned up every 24 hours. The amalgam and skimmings are ground with the addition of quicksilver in the clean-up pan to soften and clean the amalgam. The surplus mercury is expressed and the balls retorted with those from the general clean-up. Ten to fifteen minutes are required to clean up each battery, the stamps being hung up and water turned off. The general clean-up is a fortnightly or monthly affair. Two batteries are hung up, the outside battery-plates and screens removed, and inside-plates and dies taken out. The inside-plates are laid over the sluice-plates and the amalgam scraped off, taking care not to scratch the copper. Where the outside-plates are not removable, they must be protected, after they have been cleaned, by boards laid across them before attempting to clean up the mortar. The linings of the mortar and dies must be carefully washed in a tub, and scrubbed down with a brush before replacing them. If any of the castings are imperfect they should be carefully probed. The battery-bottoms are carefully shovelled into pails and removed for treatment, and after sifting out the coarser part of the uncrushed ore, the balance, consisting of several pans full of medium, coarse, and fine ore, mercury, sulphides, amalgam, pieces of iron, steel, etc., from the different batteries, are run through the last box cleaned-up. The bottoms remaining in it, when these have been finally worked down are taken out, and the bulk of the worthless material panned-off, extracting the steel and iron with a magnet; or the bottoms may be run through a tom. What remains is then put through the clean-up pan. In large mills, the barrels and clean-up batea are used. The clean-up pan should be in a special room as close as possible to the battery, the floor being of cement to prevent loss of amalgam and mercury. It should contain two cast-iron tanks, 4 feet long, 3 feet wide, and 3 feet deep to prevent loss and leakage in panning-off. It is a good plan also to give the floor a slight slope towards a small tank cemented into it, to catch any mercury accidentally spilled.

In one corner is placed the clean-up pan before mentioned, (usually about 2½ feet in diameter), provided with hard iron drags.

The final residues from the battery are ground in this pan with mercury, the refuse run off, and the amalgam collected after one or two hours' grinding.

Three men can clean up a 40 stamp mill in this way in 5 to 7 hours, at which time advantage is taken of the stoppage of the stamps to make needed repairs, and to change shoes, dies, screens, etc., if required.

It takes from two to four hours to retort the amalgam in the silver retorts used in large mills. Small mills use the cup-shaped retort. The resulting bullion is weighed and the gold melted in plumbago pots, taking 1 to 2 hours. Borings are then taken, or the bar is chipped in several places, and several bullion-assays are made to determine its fineness.

Grade of the Plates.—The inclination given to the inside copper plates is very variable; for the outside plates, it varies with the sulphides in the ore, the amount of water used, and the fineness or coarseness of the gold. Sufficient grade is necessary to prevent the pulp settling on them. High sulphides and coarsely crushed ore requires a maximum grade. The frame supporting the plates, should therefore be constructed so as to admit of adjustment to suit the ore treated.

The usual grade of outside plates is $1\frac{1}{2}$ to 2 inches per foot. The apron-plate is generally given a grade of from $\frac{1}{2}$ to $1\frac{3}{4}$ inches per foot, $1\frac{1}{2}$ inches being about the average. The sluice-plates have usually a fall of $1\frac{1}{4}$ to $1\frac{1}{2}$ inches per foot. The end of the sluice-plates should be furnished with a mercury-trap. With most ores, steep grades and a minimum of water are to be preferred to lessening the grade, and using more water.

By this plan the gold is rolled, rather than swept along by the water, and better contact is secured.

Shaking-plates suspended on movable springs are an important adjunct to the system of amalgamation; they should be the same width as the sluice-plates, but set at a slightly less grade; the movement of the ore, being assisted by the longitudinal shaking movement given by the eccentrics which actuate them.

There should be one or two drops from 2 to 3 inches in height in the line of plates; a crater-like deposit of amalgam accumulating where the pulp drops from one step to another.

The frames of the plates, with the exception of the apon-plate, which is sometimes supported on a casting bolted to the mortar,* should rest on

* If the more common method of supporting the apron-plate like the lower ones is adopted, the end board ought to be brought well under the lip of the mortar, but kept from direct contact with the battery-blocks and the under side of the box.

bearings independent of the battery framework, to avoid the jar which would otherwise ensue. The plates are held down by wooden cleats at the side.

Mercury is charged every hour or so into the mortars, the quantity depending on the grade of the ore and physical character of the gold. Finely-divided gold requires more mercury than when it is coarse. Sulphide ores also require larger charges, as ores of that class have a scouring effect on the plates and carry off the quicksilver.

The condition of the outside-plates is a guide as to whether too much or too little is being used. The amalgam should be sufficiently pasty to adhere to them, but neither too dry nor fluid enough to roll off.

From 1 to 2 ounces of quicksilver ought to be added to the battery for each ounce of gold contained in the ore. The value of quicksilver fluctuates considerably.

The loss of quicksilver per ton of ore is very variable where amalgamation takes place in the battery. Sulphide ore, especially such as carries galena or arsenical pyrites, occasions a large loss of quicksilver. The large losses in the early attempts to amalgamate the mispickel ores of Marmora, Canada, is a striking case in point.

Loss of gold accompanies the quicksilver, but its amount is liable to be overestimated by some mill-men. The loss of mercury at the Empire and North Star mills is said to exceed that of any mills in California, but *per contra* there are few mills that save such a high percentage of the gold in the ore. Their loss is reckoned to be often as great as 1 ounce of mercury per ton of stone crushed. In other instances it varies from $\frac{1}{3}$ to $\frac{3}{4}$ of an ounce per ton; $\frac{1}{2}$ an ounce per ton or 1 lb. per 32 tons of stone crushed is about a mean. The general causes of this loss are referred to elsewhere.

Concentration.—The pulp in most Californian mills is conveyed from a distributing-box at the end of the copper-plates through 2 inches pipes to the concentrators. These have various devices for saving amalgam, free-gold, and quicksilver, that may have escaped the preceding appliances, as the pulp delivered to the concentrators ought not to carry these substances. Any quicksilver, lost by previous careless handling or imperfect amalgamation, that finds its way to the belt-concentrators is lost by volatilization in roasting the concentrates preparatory to treatment by chlorination, which is generally practised in California; and if the gold be coarse, whether it be free or amalgamated, it introduces difficulties into the chlorination treatment, hence the necessity of saving these substances before they get to the concentrator-belts.

There is no preliminary sizing of the particles of ore in Californian gold-mills, and therefore the conditions essential for perfect concentration are wanting. Despite this serious disadvantage some of the concentrators, such as the Frue-vanner* and Embrey, succeed in obtaining clean concentrates with but little loss of auriferous sulphides. Other concentrators in use are the Golden Gate, Duncan, and Hendy pan. The Frue-vanner and Triumph are, however, the most popular. Each has features of superiority, and each its advocates. Both have quicksilver and amalgam-saving devices.

The construction of these machines is too well-known to require to be detailed here. The author will only allude to one or two special points that may be noticed in their arrangement.

At the Empire mill a special arrangement is used for removing the sulphides from the belt and depositing them conveniently for shovelling into wheelbarrows. The specifically lighter particles are carried down the surface of the belt by the current, and pass as tailings to blanket-sluices outside the mill. Fine particles of heavy specific gravity have a tendency to cling to the belts, which promotes their recovery.

There should be a separate tank outside the mill into which the overflow from the tanks below the belts is conducted, so that the fine sulphides in suspension in the water may not escape. The consistency of the pulp and its even distribution across the belt must be carefully regulated. The speed at which the belt travels is also very important, varying from 3 to 12 feet per minute in the Frue, and 3 to 4 feet in the Embrey. The grade given to the belts should be properly adjusted, and the surface must be level cross-ways.

The clear water jets which dilute the pulp require $\frac{1}{2}$ to 1 gallon of water per minute. The depth of pulp or load on the belts is from $\frac{5}{16}$ to $\frac{1}{2}$ inch, and the feed-water amounts to 1 or 2 gallons. The capacity of the machines varies with the character of the ore; ores carrying a high percentage of sulphides, or mineral in a fine state of division (slimes) require more concentrators than when they are of an opposite character.

Two concentrators are the usual allowance for a battery of 5 stamps. A floor-space of 20 by 10 feet should be provided for each concentrator, and if a double row be used, they should be set head to head in front of the batteries, with a passage-way of 5 to 6 feet between them.

A point of the utmost importance is to secure intelligent supervision and regularity in running, and, to do this, power to run them should be

* The writer believes he may claim to have run the first automatic feeders and Frue-vanners which were operated successfully in a North Queensland battery.

supplied by an independent motor; for when the concentrators are connected with the main driving shaft, the stoppage of other machinery (such as a rock-breaker), has a disastrous effect on the work they do.

The number of oscillations given to the machines is likewise of consequence. With the Frue, it varies from 180 to 200 revolutions per minute, with a throw of 1 inch. In the Embrey, it is 230 revolutions. Three men, one head concentrator, and two assistants can easily attend to sixteen concentrators, *i.e.*, for a 40 stamp mill per 24 hours. The duty of the former is to supervise, repair, and oil the machines while the assistants rake out and remove the sulphides to the sulphide room.

In a large mill with 80 stamps, it is preferable to employ one man per shift to attend solely to the adjustment of the machines, and an engineer to make repairs for all the machinery about the mill. One roustabout can remove the accumulated sulphides. Adjoining the concentrating-floor, on a level with it and on the sunny side of the mill when practicable, there should be a room to store the sulphides, with a concrete floor to drain off the water to a central tank towards which the floor has a slight slope. It should be well lighted so as to dry the concentrates in the sun.

The concentrates in California are generally treated by the Plattner chlorination process (unless the mill is large enough to have chlorination works of its own) at custom's establishments. These establishments charge about £4 3s. 4d. per ton for treatment, and guarantee a return of 90 to 92 per cent. of the assay value of the ore.

Blanket-sluices.—The length of the blanket-sluices outside the mill is governed by the value of the tailings. From 100 to 200 feet usually suffice. These sluices have a grade of 1 inch to $1\frac{1}{4}$ inches per foot. The sands collected on the blankets are generally ground in a pan, with a diameter of $3\frac{1}{2}$ to 4 feet, like a silver-mill pan. The pulp leaving the sluices constitute the mill-tailings, and to keep a proper supervision over the work, samples should be taken of those several times during the day and night, and their value ascertained periodically.

Mill-men are, on the whole, exceedingly remiss in this respect. An automatic sampler ensures the work being done properly. The McDermott and Starr machines are used for this purpose. The pulp is assayed three times a week, the samples being examined carefully to see to what cause the loss is to be ascribed. Microscopical examination of the sands should be made occasionally to ascertain if a perfect liberation of the free-gold from the gangue has taken place. Thorough and regular sampling is a check on the mill work, and causes the men to be alert and zealous in discharging their duties.

The wooden batea and miner's gold pan are used for panning-off.

The average value of the ore milled is represented by the value of the gold saved plus the value of the sulphides caught, and the value of the tailings. The percentage of gold saved by the mill is calculated from these factors, and represents the efficiency of the process. Other things being equal, it will vary with the class of ore, being less with brittle sulphides and fine-gold than when the opposite more favourable conditions obtain.

Most of the loss occurs through loss of sulphides, consequently a large percentage of rich sulphides, is liable to produce rich tailings. There are few ores in California, however, from which 80 per cent. and upwards of the assay value, cannot be extracted by careful mill-men in well arranged mills. The majority save 75 to 85 per cent. Exhaustive investigations, extending over 18 months, at the North Star and Empire mills, show a saving of 82 to 94 per cent. The usual percentage, according to reliable semi-monthly returns, is 86 to 90 per cent. In these estimates no deduction, of course, is made for any after loss in treating the sulphides; but this is usually unimportant, as before stated.

The mill labour in a 40 stamp mill per 24 hours is as follows:—

	£	s.	d.
One man at rock-breaker, at 10s. 5d.	0	10	5
Two amalgamators, at 12s. 6d.	1	5	0
Three concentrators, one at 12s. 6d. and two at 10s. 5d...	1	13	4
	£3	8	9

The rock-breakerman also attends to the blanket sluices, and is employed in other work about the mill. Where steam power is used, two engineers and one man to pile wood near the boilers, would be required in addition to the above staff.

The cost of milling per ton in a 40 stamp mill, with a capacity of 80 tons per 24 hours, driven by water power, therefore stands as follows:—

	s.	d.		s.	d.
Mill labour as above	0	10¼	to	0	10¼
Assaying, retorting, and superintendence	0	1¼	„	0	1¼
Supplies	0	3½	„	0	5
Quicksilver	0	0¾	„	0	2
Lubricants, screens, illuminants, machinists' time, etc.	0	2	„	0	4
	1	5¾	to	1	10½

To this must be added the variable cost of water-power. The use of steam-power would add about 5d. per ton to the above estimate for labour, with ½d. additional for repairs, lubricants, etc., incidental to using steam.

An electric plant to illuminate mill and offices costs about £125, the cost of producing the light being but little beyond the cost of extra power to run the dynamo. Good illumination is very desirable in a mill.

The charge for assaying, retorting, and superintendence is based on the salary of £25 per month for a man to perform these duties in addition to rendering other services, as clerk, timekeeper, etc., about the mine; one-half his time being charged to milling, the other half to the mining costs. At some works the mine superintendent performs these duties.

The power for a 40 stamp mill is calculated as follows:—

	Horse-power.
1 Rock-breaker	12
40 Stamps	66
16 Concentrators	8
8 Shaking-tables	$2\frac{1}{2}$
1 Clean-up pan	$1\frac{1}{2}$
1 Revolving barrel and batea	2
Total	92

Ninety horse-power will suffice if the revolving barrel, clean-up pan, and batea are run when the rock-breaker is thrown out of operation.

The use of water-power, when practicable, effects a saving in cost of plant, labour, fuel (the price of wood in California ranging from 12s. 6d. to £1 0s. 10d. per cord, and $\frac{1}{15}$ to $\frac{1}{8}$ of a cord being consumed per ton of ore crushed), repairs, and lubricants, decreases the liability of fire, affords a ready means of extinguishing conflagrations (reducing the insurance premium), and is more constant than steam. Where sufficient head is available, especially in dealing with a limited volume of water, hurdy-gurdy wheels, such as the Pelton, which develops 75 to 80 per cent. of the theoretical power of the water, and under the most favourable conditions even several per cent. higher efficiency, are to be recommended. For low pressures Leffels turbines may be advantageously adopted.

Copper-plates.—Nothing in milling practice gives greater trouble than keeping copper-plates free from yellow stain—electro-plating is beneficial; and another way out of the difficulty is to substitute Muntz metal for copper as a material for the plates, providing the ore is not excessively acid or of very high grade.

Muntz metal is an alloy of three parts of copper and two parts of zinc, a small proportion (less than $\frac{1}{100}$th) of lead being commonly added to it. Plates of this kind can be kept in good condition by using a weak solution of sulphuric acid, and are supposed to set up a slight galvanic action,

which keeps the mercurial surface in good condition. If Muntz metal be unprocurable all one need do is to take a metal that is positive to mercury, such as iron, and attach a strip of it in contact with the copper, at the head and down each side of the plate. This forms a weak galvanic couple, and in this way the oxidation being transferred to the more positive metal, the amalgamated surface of the copper will be protected from any acidity in the ore. In preparing copper-plates, advantage is often found in coating them at the start with silver, or better still, gold-amalgam, and the electro-plated copper-plates before alluded to are frequently used. There are different methods of amalgamating ordinary copper-plates, and the matter is of importance, as it has been found that when prepared with nitric acid they seem most liable to tarnish. A plate prepared with cyanide of potassium remains bright longer, whilst one which has been brought into condition with zinc-amalgam appears to stand better still.

The best results seem, however, to be obtained by using mercury in which a little cadmium has been dissolved, which can be advantageously used in small quantities both on the apron and in the mortar.* The vibrating amalgamated copper-plates, of which mention has been made to which an adjustable pitch can be given, mark a decided advance in battery amalgamation.

Copper-plates for amalgamating should not have been hard-rolled. To make them properly porous they ought to be annealed before use, heating the whole surface uniformly, and taking care that it does not get oxidized. They should weigh not less than 3 lbs. per square foot, and, within reasonable limits, the heavier they are the better. When laid smooth and true, the first dressing may be applied, by first scouring well with sand and wood-ashes, or they may be rubbed with fine emery powder, and washed with soda to make them perfectly clean and bright. After this, wash with clean water, and rub over with a solution of $\frac{1}{2}$ ounce of cyanide to a pint of water. Follow with a second washing of water applied warm, to remove the excess of cyanide, and rub on a mixture of fine sand and powdered sal-ammoniac, together with a little mercury, with a piece of blanket or a hard brush. Sprinkle the plate over with mercury, and allow it to remain on half an hour, then wash off the sand.

* In some cases it is found beneficial to add about $4\frac{1}{2}$ inches of a stick of sodium, $\frac{1}{2}$ an inch square, to a flask of mercury to quicken it, but it must be done gradually in an open dish, holding the sodium in a cleft stick, as the action is very violent. The exact quantity must be found by experiment, for too much will cause great loss of mercury and gold.

Rub it over again with the cyanide solution, and add as much more mercury as the plate can absorb. Silver-amalgam can be best laid on with a little sal-ammoniac, using a piece of rubber belting.

Narrow mortars, low discharge, and excessive quicksilver feed (which produces a fluid amalgam) decrease the percentage of the amalgam caught in the battery boxes.

Too thick a layer of hard amalgam should not be allowed to accumulate on the plates permanently. To remove it, the plates may be occasionally immersed in boiling water until it is sufficiently softened to be easily scraped off. This is better than the ordinary method of sweating them.

The sweating of the outside battery-plates and aprons of a 20 stamp mill, after running for 18 months on £3 15s. ore (notwithstanding the fact that the plates were daily and carefully cleaned), has been known to yield £4,000 worth of amalgam.

It is therefore only the "tenderfoot," or the "new chum," who would sell his old plates, much less accept the philanthropic offer of new plates for old ones (unless he had first previously sweated them himself, and then it is doubtful that he would be the gainer), unless they were thoroughly worn out.

A convex curve formed by reducing the pitch of the fixed plates will sometimes have a beneficial effect and save gold-amalgam, which would otherwise be lost. This was proved by experience at the Spanish mine. It appears a mistake to follow up the aprons with narrow-plated sluice-boxes with the object of catching fine gold. It would seem more rational to diminish the velocity of the current by spreading it over a wider surface of smaller length. In Australia, it is a common practice to arrange the aprons in steps, with a fall of 2 or 3 inches at the head of each, the pulp falling vertically on to the coppers through an iron grating made slightly convex, which is punched with $\frac{1}{4}$ inch holes, and is supported across the head of the apron, so as to catch the discharge from the step above. The amalgam collects in thicker ridges below this grating, than on the lower surface of the plate.

No one who has been through different gold-mining camps can escape noting the fact that a modern stamp mill fully equipped with improved labour-saving appliances, and arranged so as to give the maximum of efficiency and economy, is comparatively speaking a rarity. There is more difference between the work done and results obtained, from a badly constructed and carelessly arranged stamp mill and those from a model battery, than there is between the latter and some other perhaps more efficient type of reduction process.

Day by day, milling becomes more systematically managed and scientifically worked. There is a tendency towards an increased use of wire-gauze screens in wet-crushing mills in some parts of California. They ought to give a higher duty, as the area of the openings is larger in proportion than in punched plates, and they show no tendency, as was formerly thought, to amalgamate the brass wire, which would destroy and choke them rapidly.

The necessity has been pointed out elsewhere of determining how much gold is being lost in the tailings, and an examination should be made as to the manner and cause of loss, with a view to seek a remedy.

The method of conducting such an investigation is as follows:—

1. Obtain a fair average of the daily tailings sample. Pan this down carefully to ascertain if free-gold, amalgam, or mercury is escaping. If there is an apparent loss the man in charge should be either replaced or instructed in his business.
2. A quantity of the average sample is next sized by passing it through, say 60 and 100 mesh wire screens, and each of the three sizes resulting, is weighed to determine the relative proportion of each, and assayed separately. If the sample which does not pass the 60 mesh screen, assays appreciably more than the finer sizes, the loss is evidently through enclosure in the particles of gangue; in other words through the ore not being reduced fine enough; and may be owing either to the free-gold or sulphides present. The sample should therefore be pulverized finer in a mortar to see whether it is to be ascribed to the one or to the other possible cause. The result will show whether finer crushing is likely to be productive of gain by amalgamation. If the assays of the three samples are nearly alike, finer crushing would clearly be disadvantageous, reducing the capacity of the batteries and promoting liability of loss. If the pulp passing the 100 mesh sieve goes highest, a coarser screen may be tried as likely to give increased capacity without extra loss.
3. A weighed portion of the original sample should next be panned-off to determine the percentage of sulphides present. The sulphides collected are then assayed to determine their value for subsequent treatment. In this connexion, test 1 must have been previously made, as any amalgam would, if present, lead to an entirely erroneous conclusion in estimating the value of the concentrates collected.

4. Note the loss of fine-gold in the slimes, as shown by the assay of the ore screened in the second test, and determine as nearly as possible by careful panning and assay of the concentrates after amalgamating the free-gold by hand, what proportion of the loss is due to the combined and what to the free-gold.
5. If the loss is high both in the slimes and coarser sizes, the gold is probably in a condition not susceptible to amalgamation, and if after trying the remedies that will be presently pointed out, no better results ensue, a small pan test may be made in the laboratory. In searching in this direction for a more perfect result the extra profit must, however, be carefully weighed against the additional cost.
6. Assays and analyses of the gangue and its component minerals, and a microscopical examination of the actual character of the gold, may in some cases give valuable assistance in enquiries of this nature.

The causes of loss have been stated in the earlier part of this paper as being due to (1) floatation; (2) enclosure in particles of gangue; (3) inaptness of the gold to amalgamate; (4) impure mercury; (5) bad condition of the plates; and (6) by its escape with mercury.

The loss from the first cause will generally increase, as pointed out by Mr. C. H. Aaron in the *New York Mining Journal*, August 10th, 1889, (*a*) with the fineness of the gold; (*b*) in absolute quantity, though not in percentage, with the richness of the rock in gold of this latter description; (*c*) in percentage, but not in absolute quantity, with the poverty of the rock in fine-gold; (*d*) with the quantity of water used; (*e*) with the muddiness of the water, hence a medium must be found in this respect so as to dilute the pulp as far as possible without running the danger of sweeping it away with too strong a current, (it is particularly in this connexion that shaking plates are of service); and (*f*) on the degree to which coarse particles of gold and amalgam are abraded and comminuted by the stamps, which is a strong argument against amalgamating as much as possible in the mortars using fine screens, with high discharge, and a minimum of water in the battery, except under special circumstances that justify it.

The loss from the second cause (enclosure in the particles of gangue) can only be remedied by finer crushing. There is a limit to this being done, however, as the finer the stone is crushed the finer will the particles of gold be reduced as well as the rock, the smaller will be the output of the battery, and the gold obstructed by the fine particles of gangue with which the water is surcharged, will have less opportunity of settling and

amalgamating. In such a case the rock may be submitted to two distinct operations of crushing and amalgamation with an intermediate separation of the slimes, but this entails extra expense which must be set against the additional saving effected.

At the Plumas Eureka mill, the tailings of the battery, as a case in point, are taken up by Italians (who pay a royalty for the privilege), and passed into wide shallow sluices which retain the sand while the slimes escape. The sands are then ground and amalgamated in arrastras.

Inaptness of the gold to amalgamate, the third cause of loss, may be overcome chemically or mechanically. In the latter case, by passing it through a machine such as a pan or mill, in which it is ground and brightened, or by warming the battery water if the inertness of the amalgamation be due to cold. If, however, this difficulty is of a chemical nature, advantage may be found in allowing a small stream of potassium cyanide to trickle from a tank into the battery, but care must be exercised in its use, or a fresh source of loss, from gold being carried off in solution, is liable to arise.

A little red oxide of mercury dissolved in the potassium cyanide will cause every particle of free gold exposed, to be instantly coated with quicksilver, and is efficacious where other means fail, but the use of such reagents is costly.

The precipitation of lead or copper in the mortars may be prevented by adding soda or milk of lime to the water used, or by causing it to flow over broken limestone, if the difficulty be merely in the water itself.

As regards impure mercury, the fourth cause of loss, the presence of lead, copper, mercurous oxide, sulphur, etc., is admitted to be injurious. Sometimes an ore contains sulphate of lead, which being to some extent reduced by the iron of the battery, will amalgamate with the quicksilver. In a similar manner soluble salts of copper either in the ore or the water cause a precipitation and amalgamation of the copper, which, though less harmful than lead, is still injurious. In all such cases the quicksilver after being strained should be purified before re-use.

To remove small quantities of copper and lead from mercury, retorting is not necessary. Such impurities may be eliminated by keeping the mercury for some hours in an enamelled pot, under dilute nitric acid (which is best warmed), and occasionally the mercury should be stirred. The acid will dissolve the copper and lead until it becomes saturated. It may also dissolve some mercury, but this will be deposited again when a fresh lot is treated, or it can be recovered by immersing a strip of copper in the liquid.

As the amalgam removed from the plates is liable to contain a little copper, it is well to always keep it in stock under acid. When wanted for use it should be washed with clean water. Oxygen, sulphur, and chlorine may be removed from quicksilver by adding a little sodium amalgam, avoiding an excess. Lead is not wholly removed by retorting, unless the quicksilver be covered to a depth of an inch or two with powdered charcoal.

If a solution of potassium cyanide in which red oxide of mercury has been dissolved is used in the pans (as is sometimes done, with the object of facilitating amalgamation), a little zinc-amalgam should be added to the pan towards the close of the operation, to avoid loss of gold in solution.

With regard to the fifth cause of loss, Mr. E. B. C. Hambley, in trials made at one of the Indian mills, in 1886, by substituting efficient for inferior plates, and looking after them properly, succeeded in saving 63·23 per cent. of the total gold caught, as compared with 33·33 per cent. yielded previously.

When ore is crushed dry there is great danger of loss, owing to fine particles failing to become thoroughly wetted, in consequence of which (although specifically heavier) they will often float for a long time on the surface of the water.

The sixth cause of loss, that of quicksilver carrying gold, may often be caused in the same way by flowering, owing to its attrition in water with sand, powdering it to an almost impalpable dust. The loss in itself may not be a serious item, but when charged with gold it is one of some moment, as it has been proved at the North Star mill, Nevada County, that £10 8s. 5d. worth of amalgam escapes the aprons and sluices which would be lost but for the shaking-aprons and concentrators below, intervening between the tailings-race and the battery.

Another loss of mercury occurs in handling, and a third in retorting the amalgam, a portion remaining unexpelled from the bullion, and some escaping condensation. At the Keystone mill, 10 ounces per month are lost in this way. Minor sources of loss are oxidation, sublimation (extremely slight at ordinary temperatures), combination with base metals in the ore, and adhesion to metallic particles in the gangue.

What is true in regard to not cleaning the plates too frequently does not apply to the battery, as it is generally beneficial, more especially if the ore contains coarse gold, which is caught in the box, to clean it out every two or three days, notwithstanding the loss of time.

The common practice of running for a month without a clean-up of the boxes, is in such cases bad, as the amalgam runs a great chance of being flowered and thrown out of the mortar, resulting in loss. Steel tappets and

cams are coming into general use, and it is an advantage to counterbore them. Split cams are convenient for changing if a breakage occurs.

Though the hardness of quartz as tested by scratching is nearly uniform, the facility with which it can be crushed depends on its texture, *i.e.*, its friability or compactness. The smaller the rock is broken, to a uniform size before it goes to the battery, the lower and more uniform will be the feeding, admitting of lower and more rapid drop. Hence more ore can be crushed for the expenditure of a given amount of power.

Battery Water.—The amount of water fed to the battery depends on the number of stamps in each battery, character of the ore, size of screen, and discharge of the mortar. Clayey and highly mineralized ores require a maximum amount, and a roomy, long or double-discharge mortar, more than a narrow single-discharge type. The quantity used per ton of ore stamped in California runs from 1,000 to 2,400 gallons. The mean amount is about 1,800 gallons. From $\frac{3}{4}$ to $1\frac{1}{2}$ miner's inches per battery should be provided; a miner's inch is about 16,800 gallons supplied per 24 hours.

Practice in Colorado.

The following interesting particulars, chiefly taken from a series of papers on variations in the milling of gold ores by Mr. T. A. Rickard, which appeared in the *New York Mining Journal* in 1892 and 1893, illustrate the differences that obtain in milling practice in Colorado, Australia, and New Zealand.

At the Hidden Treasure mill, the stamps of the old batteries are furnished with screw tappets which have given place in the newer ones to the gib-and-key method of attachment. The stamps weigh 550 lbs., fall 30 to 32 times per minute, and drop in order 1, 5, 2, 4, 3. Each stamp makes from $1\frac{1}{4}$ to $1\frac{1}{2}$ revolutions at each fall, depending on the amount of grease in the cam shaft. The discharge, measured from the top of the die to the bottom of the screen, is 13 inches when new dies have just been put in, increasing as they wear down to a maximum of 15 to $15\frac{1}{2}$ inches.

The shoes are $5\frac{1}{2}$ inches deep, and 8 inches in diameter. The dies are plain and cylindrical, fitting into a round seat in the mortar bed. They are $3\frac{1}{2}$ inches deep, and slightly wider than the shoes, and are kept in place by being tightly packed round with tailings. The shoes weigh 83 to 86 lbs. each, the dies from 46 to 48 lbs., both being made of locally manufactured cast iron. The wear of the shoes is 11·3 ounces of iron per ton of ore crushed, that of the dies 4·5 ounces. At present fifty heads are employed on custom work, and these crush faster than the twenty-five fed with mill-rock from the California mine.

COMPARATIVE TABLE OF GILPIN COUNTY MILLS.

Name of Mill.	Hidden Treasure.	Gregory Bobtail.	Randolph.	New York.	Prize.
Number of stamps ...	75	125	50	75	25
Weight of each stamp (lbs.)	550	550	500	600	500
Number of drops per minute	30 to 32	27 to 30	30	26	28 to 30
Height of drop (inches) ...	16 „ 18	16 „ 18	16 to 18	18 to 20	15 „ 17
Depth of discharge at issue (inches)	13 „ 15	11 „ 13	14 „ 16	13 „ 15	13 „ 16
Capacity per stamp head (tons)*	1·14	1·04	·93	1·07	0·80
Capacity of entire mill (tons)	85	130	48	80	20
Size of screen (No.) ...	1½	1 & 2	1½	1½	1½
Description of screen ...		Burr slot	alternate	punched.	
Percentage of concentrates per ton of ore	13	14	20	15	12
Value of concentrates per ton	£3 2s. 6d.	£2 1s. 8d. to £5 4s. 2d.	£2 1s. 8d.	£1 9s. 2d. to £2 1s. 8d.	£2 1s. 8d. to £5 4s. 2d.
Percentage of bullion obtained in retorting ...	40	40	33 to 47	40	35
Fineness of bullion (per 1,000)	782 to 786	800 to 850	750 „ 850	750 to 800	750 to 775
Life of the screens (days)	81	60	16	25	75
Loss of mercury per ton of ore (dwts.)†	4·3	5·2	9·8	3·7	9·7
Consumption of water per stamp per minute (gallons)	2	2·3	1·4	1·3	1·5

Custom milling is a great feature of the mills of this section. The charges are £1 11s. 3d. per day per battery of five heads, including the concentration of the pyrites and their shipment on the railway cars. For small lots the rates are £5 4s. 2d. per cord for milling and 8s. 4d. per cord for concentrating. This last charge varies from 4s. 2d. to 12s. 6d. according to the percentage of pyrites present.

The cord of mill-rock above alluded to is equal to 7½ to 8 tons, while one of smelting ore averages 9 to 10 tons. The screen is of the burr-slot description, the slots being horizontal and alternating. No. 1½, equal to a 50 mesh wire screen, is usually employed. The screen surface is 4½ feet by 8 inches, 200 feet of screens being used up in a year equivalent to 66·6 screens annually. The average life of a screen is therefore 81 days. With the ore from the California mine they last three months by turning the lower portion, which wears most rapidly, to the top. The amalgam

* 2,000 lbs. each.

† Mercury is sold by avoirdupois weight—a bottle contains 76½ lbs.

yields 30 to 50 per cent. of bullion, which contains in addition to the gold, 207 to 211 thousandths of silver. Six and a half bottles of quicksilver are lost annually, representing 4·3 dwts. per ton of ore crushed by 75 stamps.

Most of the gold is caught in the mortar-box, which is effected to a slight extent by the free mercury added, but chiefly by two amalgamated plates arranged along the front and back of the box. They are both of plain copper, both 4½ feet long, the back one being however 12 inches wide, the front one 6 inches. The former is set at an angle of 40 degs., the latter nearly vertical.

The front of the battery above the screen frame is covered with canvas, by lifting-which the mill man can introduce his arm, and tell by the feel of the front-plate whether the right quantity of mercury has been added by the feeder. This saves stoppage of the battery and removal of the screen. On the average, the feeder adds half a thimbleful of quicksilver every hour. A test has shown that in crushing 8 tons of ore, carrying ½ ounce of gold per ton, 4½ ounces of mercury were added. After the first six hours a drop as large as a medium sized pea added every hour sufficed.

The gold-saving appliances following the battery are amalgamating tables, blankets, and concentrators. The first of these are 12 feet by 4 feet in one length, covered with copper, with a grade of 2⅛ inches per foot, which is greater than in California.*

In crushing 24 tons of ore carrying ½ ounce per ton, it was found that one such table required 5 ounces of mercury to dress it, whilst 3 ounces were used in dressing the front inside-plate, and 4 ounces for the back one.

The blanket-strakes or strips are 3 feet long and 18 inches wide. They are washed three times per 24 hours and serve to arrest any escaping amalgam mercury, rusty gold and the heaviest pyrites, together with particles of stone to which gold is still attached. Unless to save this last class of material, concentrators could replace them without loss. From the blankets the pulp passes to concentrators known locally as bumpers, a variation of the Rittinger.

In the Hidden Treasure mill there are 5 of these tables. The speed is regulated by the percentage of pyrites present, averaging 130 strokes per minute. Of the total amalgam obtained, ⅔ is yielded by the inside-plates. On cleaning-up, the sand found in the battery round the dies is not panned but returned to the mortar. The outer tables are cleaned-up every 24 hours, but the inside-plates only every 48 hours. With poor

* Page 114.

ore the last named period is prolonged. At the general clean-up the amalgam from the plates is placed in a mortar and ground with hot water till of even consistency, the dirty water being decanted and mercury afterwards added to thin the amalgam.

Thus prepared it is run from one porcelain dish to another several times, and as the dirt and pyrites rise to the surface they are skimmed off by hand. The clean amalgam remaining is pressed through canvas. The skimmings resulting are re-introduced into the mortar and re-ground with fresh mercury and hot water. When fairly clean, a bit or two of potassium cyanide is added to render the mercury more lively.

In retorting, the retort is either chalked or lined with paper. The balls of dry squeezed amalgam are introduced into the retort, broken with an iron rod, and pressed down till hard and uniform. The cover is then put on and luted down with clay. Fine pan-sludge is often used in Australia, and forms a good lute. The only chemical used is potassium cyanide. Of this twenty-six 10 lbs. canisters were used in a year in the treatment of 28,793 tons of ore crushed. The tables are dressed every 12 hours with a weak solution containing 2 ounces dissolved in 3 gallons of water. The tables are brushed twice daily with a mop, mercury being sprinkled over them if the amalgam be too dry. The feeding is done by hand, and there are no rock-breakers. Feeders are paid 12s. 6d. per shift, and there is one for every 25 stamps. The mill is worked four months in the year by water power, four months by steam, and four months by both combined.

Firewood costs 19s. 9½d. per cord delivered. The cost of milling in 1890 was at the rate of 3s. 6d., but in 1891 it was decreased to 3s. 3d. per ton. The labour employed is thus distributed per month of 30 working days, 75 heads. One mill-man £36 9s. 2d., 1 assistant £20 16s. 8d., 6 feeders (3 per shift) at 12s. 6d. per day, £112 10s. per month, 2 concentrator men (1 per shift) at 12s. 6d. per day, £37 10s., total £207 5s. 10d., or 1s. 7d. per ton. Twelve hour shifts are worked.

While all the mills of the district are much of the same type and are engaged in crushing ore of much the same character, there are slight differences of detail in them. The stamps are all of light weight, rendered necessary by their high drop, which would be impossible with stamps of 850 to 900 lbs. The speed is also directly affected by the same cause, for the work required to lift the heads to a height of 16 to 18 inches prevents the rate of fall exceeding 32 drops per minute for good work, 40 is probably the practical limit, but the present tendency is in favour of keeping

it up to a speed of 32 drops, rather than running slower. The New York mill has the longest drop, but in this respect it follows the older practice.

The Gregory Bobtail mill fairly indicates the construction of the most recent plants in regard to depth of discharge. The precentage of concentrates obtained from the ores of this district ranges from 12 to 20 per cent., showing the refractory character of the mill-stuff. The value is very low, averaging £2 1s. 8d. to £2 10s. net per ton. The concentrates and blanketings undergo no further treatment, and are shipped direct on railway cars from the doors of the mills to the smelters at Denver. The freight is 6s. 3d. per ton, and cheap concentration supplemented by light smelting charges, alone makes the treatment of the concentrates profitable. Ninety-five per cent. of the silver and gold contents as fixed by assay is paid for, £1 13s. 4d. being deducted per ton for treatment; formerly a minimum rate of £1 10s. 2½d. was allowed, when this class of material was less plentiful. The retort percentage depends on the coarseness of the gold and the thoroughness with which the amalgam is squeezed, imperfect manipulation causing a difference of as much as 10 per cent.

The small capacity of the mills, the slight use made of the blankets, and the high slope of the amalgamating-tables (1½ to 2⅛ inches per foot), considering the deep discharge, account for the comparatively small amount of water used, viz.: 1½ to 2⅓ gallons per minute per stamp. The screens are of local manufacture, and are made of planished iron (an imitation of Russian sheet), size No. 24. The openings are straight slots set alternately. Nos. 1, 1½, and 2 most in use, are considered equal to 60, 50, and 40 wire mesh; they have nothing like the same discharge-surface however; a large proportion of the pulp being kept in the box till it would pass a 100 mesh wire screen. Their use is only justified by the main idea of Gilpin County practice, that of retaining the pulp inside the box till it is amalgamated. The side which carries the burr edge of the punched openings is always placed facing inside, to break the pulp and prevent choking.

There is a wide difference, as will be seen from the table, in the life of the screens in the different mills—far greater than can be explained by the greater or less attention of the mill-man, and the extent to which he is willing to allow the screen-slots to be enlarged by wear. Mr. Rickard accounts for this by their order of succession of the mills along the creek which supplies them with water. The Hidden Treasure mill receives water comparatively clean, and after having used it returns it to the

creek with a certain percentage of sulphuric acid, derived from the contact of the water with the pyrites. It is not surprising therefore that its screens last the longest—81 days. The Prize mill, in the same way, adds its quota of sulphates to the water, to the additional injury of the Gregory Bobtail screens, which only last 60 days.

The lives of the screens of the New York and Randolph mills are measured by days (25 to 16) instead of weeks, owing no doubt to the additionally corrosive action of the water lower down, due to the acid waters which issue from the underground workings of the Gregory mine above them. Taking the Hidden Treasure as most truly typical of the life of the screens in this district, we find 432 tons of ore pass through a grating before it is worn out. At Grass Valley, a screen will live to pass 200 tons, and at Bendigo (Australia) 134 tons. The very roomy character of the Colorado mortar-box probably accounts in part for this favourable comparison.

The loss of mercury is greatest in the mills with the deepest issue, which bears out what has been said elsewhere as to the danger of stamping it to flour, *i.e.*, subdividing it into minute globules, which are liable to become coated with pyrites or foreign matter, and are then borne away by the water. The variable loss of mercury at different mills may also be largely explained by the larger number of clean-ups where more lots of ore are treated coming from different mines, which lead to additional manipulative losses.

The ore of the California mine, treated at the Hidden Treasure mill, is representative of the ore of the district. It consists roughly of 15 to 20 per cent. of quartz, and 60 to 70 per cent. of vein-filling (other than quartz), an altered form of the rocks enclosing the vein. As these latter consist of gneiss, alternating with granite and mica-schist, the gangue is largely felspathic.

Of the metallic constituents of the ore, iron and copper pyrites predominate; grey copper (fahlerz or tetrahedrite) arsenical pyrites (mispickel) and galena are also present in noteworthy proportions. Blende is sometimes seen and chalybite (carbonate of iron) appears occasionally. The grey copper, which is here antimonial, is generally remarkably favourable for the presence of gold, a fact which would prove a valuable index in selecting the ore, were it not so often confounded with arsenical pyrites. Quartz (especially favourable when of a blue tint) is always associated with the pyrites in rich ore. The writer may here remark that he has observed this same peculiarity of colour in the rich quartz of the Mysore mines in India.

The following results of assays made by Mr. Rickard, upon a typical piece of ore broken in the 1,700 feet level of the California mine, throws some light on the distribution of the gold and silver :—

Mineral.	Gold. Ozs. per Ton.	Silver. Ozs. per Ton.	Remarks.
Iron pyrites ...	0·65	4·85 ...	White coarsely crystalline.
Copper pyrites ...	0·85	53·50 ...	Flaky dark yellow.
Grey copper ...	0·90	38·65 ...	Chiefly covering the last.
Blende	0·16	6·45 ...	Black crystalline.
White quartz ...	3·32	7·35 ...	Opaque massive, with small
Bluish quartz ...	3·56	5·84 ...	crystals of pyrites throughout.
Flinty quartz ...	0·18	1·90 ...	Brown vitreous.
Felspathic gangue	0·90	2·35 ...	Soft granular white.

This analysis bears out the experience of the mills, half the gold-contents of the being are extracted by the first amalgamation in the mortar box. The gold cannot therefore be chemically combined with the pyrites. On the other hand, the more highly mineralized the ore the richer it usually is also. There is no doubt the silver-contents are for the most part associated with the copper-bearing minerals, while the gold is enclosed in the quartz, especially when that quartz is immediately associated with pyrites. Neither blende nor galena is an attendant upon the gold, and both are a nuisance in the mill.

The following figures, representative of the output for 1890, further illustrate the nature of the ore—150·44 tons of smelting ore averaging £19 6s. 7½d. per ton net; 1,376·03 tons of concentrates averaging £3 2s. 9d. per ton net; 10,320·57 tons of mill-stuff averaging £1 10s. 11d. per ton.

The smelting ore is the high grade sulphide ore picked out at the mine and shipped direct to Denver. The mill-stuff yielded 4,766·39 ounces of bullion, worth £3 9s. 4½d. per ounce. Of the total tonnage, 90 per cent. was mill-ore, representing 84 per cent. of the total value. The mill-ore yielded 13 per cent. of concentrates.

A test made in March, 1891, to determine the completeness of the mill extraction on a lot of 8,400 lbs. of ore, which contained 4 per cent. of moisture (leaving 8,064 lbs. net) showed, after passing through a breaker and rolls (specially erected for accurate sampling), an assay value in gold of 1·85 ounces, and in silver of 8·75 ounces per ton.

The contents of the 8,064 lbs. were therefore 7·46 ounces of gold and 32·86 ounces of silver. At the smelter such ore would be worth £5 12s. 10d. per ton net (smelting charges being £2 10s. per ton and 95 per cent. of the gold and silver being returned and paid for on New York quotations) or a total value of £22 14s. 11d.

This ore was sent to the mill and yielded, after treatment for 18 hours in a 5 stamp battery, 6·70 ounces of bullion, worth £3 6s. 8d. per ounce, or £22 6s. 8d. and 2,325 lbs. of concentrates containing 15 per cent. moisture, leaving 1,977 lbs. net equivalent to 24½ per cent. of sulphides in the sample. The assay of the concentrates gave 1·76 ounces of gold and 10·37 ounces of silver per ton, or a total for the 1,977 lbs. of the above assay value, representing £9 3s. 5½d., or deducting smelting charges, £7 10s. 1½d. net.

To compare these results, the milling cost was at the rate of 3s. 6d. per ton; therefore the mill returned, after making all deductions, £29 2s. 8½d., estimated as follows:—Bullion, £22 6s. 8d.; concentrates, £7 10s. 1½d.; total, £29 16s. 9½d. Deducting milling cost at 3s. 6d. per ton 14s. 1d. equal £29 2s. 8½d.

At the smelter, the amount received, owing to the larger deductions and charges, reaches the smaller sum of £22 14s. 11d. Commercially, therefore, the mill ore paid the miner better than the smelting ore.

As a test of the mill-work the figures are as follows:—There was in the ore, 7·46 ounces of gold and 32·86 ounces of silver. There was extracted as bullion, 5·25 ounces of gold and 14 ounces of silver, and in the concentrates, 1·74 ounces of gold and 10·22 ounces of silver, or a total of 6·99 ounces of gold and 24·22 ounces of silver. Thus the mill, including the value of the concentrates, saved 93·8 per cent. of the gold and 74 per cent. of the silver.

The mill did not, however, complete the extraction of the gold and silver in the concentrates, so that it actually obtained, by amalgamation alone, 70·4 per cent. of the gold, and 42·6 per cent. of the silver.

This is said to be fairly representative of returns obtained on a large scale. Generally speaking, it has been found that the mill yields as many ounces of base bullion as there are ounces of pure gold found in the ore by fire-assay. The mill gold is 780 fine.

Considering that Gilpin County ore is probably one of the ores running highest in sulphides treated by amalgamation at the present day, the extraction of the stamp mills is certainly extremely good. That this is so, is due to the proper recognition of the necessity for altering modes of treatment in accordance with differences in the character of the ore treated—the first principle of successful milling.

The ore of the district has been described, and it may be added that in this locality ordinary panning, except with surface ores, will give no colours of gold, even with material which in the mill yields rich returns.

The *raison d'être* for the roomy mortars, slow drop, and deep discharge which characterize the Black Hawk mills are fully explained by Mr. Rickard. His views entirely agree with the writer's, expressed in a later part of this paper.

The absence of rock-breakers in the Gilpin County mills is largely due to their bad situation, which was chosen to utilize as far as possible the motive power of the creek. The position of the mill-buildings in fact (except at the expense of elevating the ore) prevents the erection of ore-bins and grizzleys, which are necessary adjuncts for running a stone-breaker economically.

Though the mills, crushing as they do very slowly, have not the same crying need for a rock-breaker as that which exists in a Californian or Australian mill, it is no doubt a defect ; for apart from the improvement in the feeding which follows the introduction of a breaker, the irregular work of the sledge must tend largely to increase the strain on the mill machinery, as evidenced by the wear and tear of the shoes and dies which in this section is excessive.

On the score of regular and accurate feeding, the automatic machine is preferable to the average man, who cannot always resist the temptation of an occasional pipe or spell of rest. Their economy is most apparent of course where stamps crush fast, but even in a case like that of Gilpin County practice, an advantage may be shown in adopting them, notwithstanding that one man feeds 25 heads.

For 75 stamps, the cost of feeding comes to a total of £1,354 3s. 4d. per annum, while, on the other hand, if a mill of the kind were supplied with the most expensive of self-feeders, the cost of the additional plant would not exceed £834 6s. 8d.

Feeding machines are, however, of little use unless preceded by grizzleys, breakers, and ore-bins, and therefore recognizing the unfortunate position of the mill-buildings at Black Hawk (chosen in the days preceding the introduction of improved labour-saving appliances), however averse one may be to methods out of date, and machinery which is incomplete, one cannot say that, from a shareholder's or mill-man's point of view, present practice could be advantageously altered.

The Black Hawk mill-man has been trained in the best school, that of experience, supplemented by the necessities of keeping a close watch over the treatment of an ore subject to frequent changes in mineralogical con-

stitution. The competition for customs ores renders him in fact careful in the treatment of the stone, and keenly awake to any possible improvement of method promising ultimate economy.

The milling is recognized to be as important as the mining, and mills are not placed under the direction of men who are simply good miners, good chemists, or anything you will, but assuredly bad and inexperienced mill-men. In Australia it is not uncommon to consign a first-class battery to the tender mercies of an engine-driver who, in addition to tending the engines, is supposed to supervise the general mill work.

In Gilpin County, the management of the mill claims an equal or greater share with that of the mine. The needs of the district have produced men who are fully conversant with the bed-rock principles of gold milling, and such men are not too well paid, though earning more than the mine foremen.

Customs milling has had the beneficial effect (by rendering the mill owners anxious to gain the confidence of the public) of placing the right sort of men in charge, and by encouraging competition in doing good work has benefited the mill-men themselves. Gilpin County practice in fact leaves the impression of good work, intelligently and conscientiously done—two factors of the first importance to the mine owner and mining industry.

Colorado's production of gold appears to have been largest in 1886, when it is stated to have produced 4,450,000 dollars (£1,250,000) worth as against 3,883,859 dollars (£809,135) in 1889, whilst its silver output was valued at 23,757,751 dollars (£4,949,530) in 1889.

Practice in the Thames District, New Zealand.

This once famous mining district, also known by the Maori name of Hauraki, is situated in the north-eastern corner of New Zealand. Though the output has now dwindled to about 30,000 ounces per annum, this has been in its day one of the richest gold-fields of the world. In 1871, the output was 330,326 ounces valued at £1,188,728. The Caledonia mine in the first 12 months' operations produced 10 tons of gold and paid £600,000 in dividends. The maximum depth yet attained is 600 to 700 feet, and the veins have unfortunately proved far less rich in depth than near the surface. The development of the gold-field has been crippled by share-jobbing, the curse of many other Australasian camps. Few districts have had so brief but brilliant a record, and few perhaps have lost such a large proportion of the gold extracted from the mines. Milling is conducted under the difficulties presented by ores of a very complex

composition, but so far the efforts made to overcome them have been of a very elementary description. It is for this reason that the tailings-mills on the field are to-day amongst the most profitable undertakings.

Briefly stated, the method of milling consists in catching all the free gold by means quite unsuited to the character of the ore, and allowing the remainder to go to enrich the sea-beaches. The character of the ore and lode-formation in which it occurs, help partly to explain a state of things which calls for such severe criticism. The bulk of the gold comes from narrow veins and extremely rich pockets traversing a decomposed andesite. Such ore-bodies must necessarily be uncertain in behaviour and of limited extent. Like the deposits of Nagyag and Veraspotak in Transylvania, to which they have a striking resemblance (the country rock, the deposition of the ore, and the character of the gold specimens, as well as their alloyage with silver, being very similar), the pockets found on the Thames are occasionally of extraordinary richness. For instance, one lot of 2 tons 8 cwts. crushed 2½ ounces per pound, and a boulder of 2½ cwts. yielded 3,500 ounces. These crushings of small quantities of very rich ore pay the dividends, and the bulk of the output being of small value in proportion to the specimen ore, the former has been sacrificed to the latter.

The chief features of the milling are indicated by the subjoined table :—

Name of Mill.	Saxon.	Moana-taeri.	Cambria.	Kuranui.	Comer.
Number of stamps	32	40	20	20	20
Weight of stamps (lbs.)	785	659	620	670	840
Number of drops per minute ...	72	66	76	70	63
Height of drop (inches)... ...	9	8	9	5½	6
Average depth of discharge (ins.)	2½	2	3	2¼	2½
Capacity per head (tons) ...	1·8	1·4	1·7	2·5	3·6
Capacity of mill (tons)	58	55	35	50	72
Description of screen		Punched	Russian	iron.	
Number of holes per square inch	148	170	180	160	160
Fineness of bullion (per 1,000) .	663	641	674	605	589
Percentage in retorting	42	40	40	45	48
Life of screens (days)	6	6	5	5½	5
Loss of mercury per ton of ore (dwts.)	14·5	15·2	—	—	—
Number of berdans	8	21	15	7	5
Number of other pans	3	4	—	—	—

No concentrates are obtained in any of these mills.

The Saxon mill it will be observed contains 32 stamps, with an extra single stamp kept for the treatment of specimen ore. The weight of the stamps, shoes and dies, and rate of drop are not the same, though the latter (subject to variations) averages 72 per minute. The depth of discharge is about 1 inch at the time of putting in new dies, increasing to

a maximum of 4 inches as they wear down. The height of drop is from 8 to 11 inches, depending on the hardness of the ore. The single stamp weighs 7 cwt. It is given a 7 inches drop, and a speed of 60 drops per minute. This specimen stamp is a curious feature of all the mills. The screen or grating used is of Russian iron, imported from Swansea. The life of a grating averages 6 days. The openings are round, punched with 148 holes per square inch. The loss of mercury is at the rate of a bottle (75 lbs.) per month, three bottles being kept in stock. No mercury is used in the mortar save for the specimen stamp. It is employed on the plates and in the ripples or wells, and in the pans. Owing to the flowering produced by the pans, the loss of mercury is excessive, viz., 14½ dwts. per ton of ore treated. The amalgam retorts about 42 per cent. The ore brought to the mill is discharged into stalls behind the batteries. The stone is there spalled and hand-fed. The ore varies in hardness according as the andesitic vein-filling is decomposed. The quartz itself is often saccharoidal. The shoes and dies of local manufacture are of white hæmatite cast-iron, the die differing from the shoe in being unchilled. The former is 10 inches in diameter and 4 inches deep, the latter 9½ inches in diameter and 10 inches high. The die is cast with a flanged footing to keep it in position. The mortar-boxes are faulty in design, being too roomy inside.

The pulp is discharged on amalgamating-tables 7 feet long and 4½ feet wide. These are in three divisions, of which the two upper ones only are copper-plated. The first length is 2½ feet, inclusive of a well 2½ inches wide. This well contains mercury. The next division is 18 inches long. The ripples (riffles) are four in number, one only (that already mentioned) containing mercury. The other three (2 inches deep) are blind-ripples. The gold saving is effected by the plates and wells, and indirectly by blanket-strakes whose residues are treated in pans. The mortar-box is merely a crusher, not an amalgamating appliance.

The plates of Muntz metal are roughly cleaned every 4 hours. The wells are of little assistance, and are really unsuitable to ores containing a notable percentage of sulphides. They tend also to conceal the careless use of mercury.* The surface of the bath of mercury is constantly coated with a scum of sulphides, which prevents contact with any gold passing over it. The wells are skimmed with a cloth every 4 hours, and

* With plates immediately in front of the battery-screens a check is afforded of the work inside, and of the proper proportioning of the quicksilver feed or the reverse, while the well hides the fact, for a time, of a too liberal use of mercury or otherwise.

the mercury placed in them is squeezed once a week. The six wells, one to each battery, catch 12 ounces of amalgam, out of the weekly yield of 200 to 250 ounces from the entire mill.

The blind-ripples are cleaned with a scoop every half-hour and merely catch the heavier sand and sulphides which go to the pans. The blankets are washed every hour, the time, more or less, varying with the richness of the ore, and proportion of sulphides. This washing is done by one boy on each shift, of which there are three. Each boy is paid £1 per week. The blanketings go to the pans. The mill contains the following pans:—

	Diameter. Ft.	Ins.		Depth. Ft.	Ins.
8 Berdans	4	6	...	0	9
2 Watson and Denny	5	4	...	2	6
1 Price (local)	5	8	...	2	7

The berdans and one Watson and Denny pan treat the blanketings. The other two pans work tailings. The berdans have a pitch of 16 inches in 3 feet 6 inches, or 1 in $2\frac{2}{3}$. The speed is 23 revolutions per minute. The amalgam is removed every 24 hours. A drag is used, which consists of a slipper or shoe weighing 196 lbs., and a top or boss of 233 lbs. weight. They are held together by a key. In Queensland two bolt-ends are cast into the shoe, and passing through the boss, are secured with nuts. The two parts are cemented together. A shoe lasts about $4\frac{1}{2}$ months.

The distribution of the amalgam in the mill at the fortnightly clean-up preceding Mr. Rickard's visit was as follows:—Plates, 223 ounces; mercury wells, 24 ounces; pans, $43\frac{3}{4}$ ounces; 35 lbs. of specimen ore, 164 ounces; this gave 203 ounces of base-bullion, which yielded 200 ounces 4 dwts. of melted gold obtained from 476 ounces of amalgam, the balance being made up with skimmings, etc. The ore treated gives an average yield of 15 dwts. per ton.

The following figures show the cost of treatment at the Saxon mill per 24 hours with 33 stamps, crushing 63 tons, including the specimen stamp:—3 feeders at 6s. 8d. per shift of 8 hours, £1; 3 boy-feeders at 5s., 15s.; 3 blanket-boys at 3s. 4d., 10s.; 3 amalgamators at 8s., £1 4s.; total labour, £3 9s. The head amalgamator is manager of the mill; the cost of labour is therefore 2s. $10\frac{1}{2}$d. per ton, while the total cost, including wear and tear, alterations to machinery, interest on capital, etc., is 4s. 1d. per ton. At the Moanataeri mill, which has a larger plant, the cost is 3s. 9d. per ton. The disparity of drop, shown in the table, is accounted for by the fact that the three first-named mills are treating ore coming from a greater depth than the others. Being harder, the stamps in the former

batteries have a fall of 8 to 9 inches as compared with 5 to 6 inches in the other two mills.*

The depth of discharge from the top of the die to the bottom of the grating is left to depend on the wearing down of the die. The importance of having a depth of discharge, suited to the particular mode of working aimed at and the particular ore treated, is a point quite unappreciated. The holes in the screens vary in number from 148 to 180 per square inch. Owing to the presence of acid sulphates, derived from the oxidation of sulphides, the life of the screens in the mills treating surface-ores is shorter than in those where harder stone is crushed. Owing to the greater speed of crushing, however, the screens in the Comer mill, for instance, last during the passage through them, of 90 tons, as against 54 tons at the Saxon mill. Their life, which is comparatively short, is due to acid mine waters, which act also, indirectly, on the partially decomposed metallic sulphides in the battery. This action of the mine waters of the district is so marked as to suggest the effects of a slowly dying solfataric action.

The loss of mercury is high (75 lbs. per month for 30 stamps), due to flowering, caused by grinding in the pans, and sickening produced by the presence of antimonial and arsenical minerals in the pulp. The bullion is of low grade, having a fineness which ranges from 589 to 674 per thousand.

The lodes, it has been said, are small and irregular; they traverse a hornblende-andesite, which is often brecciated. The lode-formation is confined to certain belts marked by the decomposition of the country rock, the mill-stuff carrying this latter in large proportion. The ore is silver as well as gold-bearing. The gangue is largely quartz, which penetrates the vein in veinlets and stringers. Sometimes the quartz is soft and saccharoidal. While the gold is frequently visible in the quartz in minute threads and particles, it is also largely associatied with copper and iron pyrites, blende, galena, etc. Silver occurs as pyrargyrite, proustite, argentite, etc. The two precious metals are found associated with tellurium as petzite and sylvanite; selenides are known to exist in the ore; and antimony, in beautiful crystals of stibnite, is often seen.

Generally speaking, the ore has a very variable hardness and composition; it contains a very large variety of metallic sulphides, and must be

* With a soft decomposed ore, or one in which the gold is very fine, a slow rate of drop is generally to be recommended. Soft and high sulphide ores should usually also be stamped with a low drop.

considered as a combined silver and gold ore, containing from ½ to 10 per cent. of sulphides with an average of 2 to 3 per cent.; it approaches the boundary-line dividing a free-milling from a refractory ore.

Reviewing the chief characteristics of the milling, we see that the feeding is done by hand and is very rough, being left to boys, instead of employing trained men. The former shirk breaking big pieces of rock, preferring to throw them into the feed-opening, where, if they stick, they are belted-in with a few blows of a sledge-hammer. The result of the absence of rock-breakers and automatic feeders in lieu of this bad hand-feeding is seen in the excessive wear of the shoes and dies, averaging 14·5 ounces for the shoe and 7·5 ounces for the die. The feeding is also too high, the batteries being kept choked with ore, which reduces their efficiency.*

The mortar-boxes are of the same pattern, whether employed for rapidly crushing soft material or the slower treatment of average ore. As no amalgamation is done inside, they are too wide. When the mortar is merely a pulverizer, the pulp should be expelled as soon as possible ; too roomy a box promotes dead-stamping, keeping in ore, after it has been stamped fine enough to escape under ordinary circumstances. When amalgamation takes place inside, the case is different, but at the Thames, it would seem expedient to use narrower mortars.

The destruction of the gratings is particularly rapid, owing to the unusual quantity of proto-sulphates of iron, copper, manganese, and alumina present ; and with wet mill-stuff which has lain long in the stopes, or on the surface, the action is very marked. As is usual in Colonial mills, the screens are fixed vertical (instead of having a slight forward inclination at the top), which tends to wear out the lower portion, faster than the upper, as well as to diminish the discharging capacity of the screen-surface ; as the ore particles are not necessarily obliged to strike an opening, to escape. The water shot against an inclined screen from the inside, has also the advantage no doubt of carrying a certain quantity of material out, in running down the inclined screen-surface, unless burred screens are used.

* An experienced millman can tell by the sound of his stamps if they are running properly. Any particular stamp can be tested by holding the stem lightly between the fingers, about 18 inches above the mortar-box, and allowing the hand to be raised and lowered, by the rise and fall of the stamp. If the stamp is overfed it will fall with a dull short stroke; if underfed, with a long vibrating, ringing blow (like a hammer on an anvil); if properly fed, with the sharp well-defined crack you ought to expect.

The mercury wells only confirm the general experience that with ores containing an appreciable quantity of sulphides, they are of little practical value, not being even of a type (like those of Clunes), which compels the pulp to pass through the mercury-bath, so far as that end is capable of attainment. The blind-ripples occupy an amount of time which appears quite out of proportion to the quantity of sulphides and heavy sand which they collect.

The blankets are washed at too long intervals at most of the mills running on company ore, and therefore, tributers (who understand this) generally attend to that part of the business themselves, washing the blankets and skimming the wells every half hour. The blankets, which are 2 yards wide cost 12s. per yard ; they last three months.

The substitution of a drag, or pair of drags, certainly is an improvement on the ball, formerly used in the berdans, which wallowing about in the mercury and amalgam (at the bottom of the pan), presented extra chances of flowering them. The drag being fixed to the side of the pan keeps the grinding and amalgamation to some extent apart, but more or less mercury is carried round constantly with the sulphides by centrifugal action, hence the evil is but partly mitigated.

Systematic attempts at proper concentration by modern methods do not appear to have been made, in consequence of which a Newberry-Vautin chlorination plant erected at the Thames had to shut down for want of being able to obtain a regular supply of concentrates. The mill results seem to show that only that part of the gold which is readily amalgamated is caught, while the silver contents escape.

The silver in the bullion is not in fact due to the amalgamation of the silver minerals in the ore, since the proportion of silver to gold in the bullion corresponds with the native gold, which like that of Transylvania, is of very low caratage.

On the most favourable estimate but 50 per cent. of the gold is saved, leaving out of account the silver. Much of the gold and nearly all the silver is carried away with the slimes, which being carried out along the foreshore have produced an accumulation of tailings, estimated to amount to at least a million tons, carrying half an ounce of bullion per ton.

That this is not an exaggerated statement is proved by the success of the tailings-mills, which pay well.

The largest of these plants contain twelve Watson and Denny pans. The tailings are elevated and conveyed to them simply and effectively by a

small hydraulic elevator, the jet being $\frac{5}{8}$ inch in diameter, with a $2\frac{1}{2}$ inches pressure pipe, and 3 inches elevator or discharge-pipe. The water used is under a pressure of 60 lbs. per square inch. The launder from the upper end of the elevator-pipe conveys the tailings to wide strakes, where the poor slime is washed away, and the rough stones are picked out, before feeding the residual material to the pans. The cost of this treatment, including insurance, interest on plant, wear and tear, etc., amounts to 3s. 6d. per ton.

Mr. Rickard points out that while blankets, followed by pans, form a process which is quite ineffectual as regards saving the silver contents of the ore, it is also badly suited to extraction of any free-gold remaining in the pulp, after its passage over the plates.

The grinding action of the pans upon the sulphides forms slimes, which sicken the mercury, cause its direct loss, and further spoil its power of amalgamating.

The ore is both silver and gold-bearing. The former metal is chiefly associated with the sulphides, the latter is mostly in the quartz, while both minerals occur combined as tellurides, etc. Some separation is therefore needed between the silver and the gold-bearing portions of the ore. He (Mr. Rickard) suggests, therefore, that from plates the pulp should pass direct to concentrators, and thence to pans. The concentrators would separate out the silver-bearing minerals and some of the combined gold, and the pulp, freed from the sulphides, would go to the pans, which would complete the extraction of any free-gold remaining.

This combination process would obviously be an improvement on present methods, without involving much expense.

The work might be further lightened and improved upon by introducing classifiers of the Spilzlütte type, between the amalgamating-tables and the concentrating-machines. The handling of the concentrates would then be by an auxiliary process, and would not interfere with the present amalgamation upon the tables.

The use of Muntz metal in place of copper, has been referred to elsewhere, and its first introduction for the purpose took place in this district in 1875. Owing to a scarcity of copper, the local ironmongers, who imported it for sheathing ships' bottoms, sold it as a substitute. Its use rapidly spread, though the sheets locally used are too thin for the purpose, being of the thickness known as No. 18. They last, however, from three to five years.

Muntz metal as applied to amalgamation has been found to possess the

following characteristics :—It does not absorb amalgam like copper, the latter requiring to become thoroughly coated before the plates will work well.

Muntz metal has very little absorbing power over the mercury, and the amalgamation is relatively to copper, very superficial. Hence the amalgam formed on the Muntz metal is more readily detached, facilitating the clean-up. A simple rubber is always sufficient, so that the use of a steel scraper is avoided.

Test crushings are more reliable in consequence, since if the copper-plates are scraped too close before putting through a fresh lot of ore, they have not a fair chance to amalgamate, while if amalgam be left on, gold is obtained which does not belong to the stone on trial. It is not so with Muntz metal, which can be readily deprived of all its previous gain of gold-amalgam without impairing its efficiency, and is therefore specially adapted to customs mills.

On the other hand, for rich ore, Muntz metal is not to be recommended, as there is insufficient body in it, that is to say it is sooner saturated with gold-amalgam than copper. In the same way silver-plated copper, will carry more amalgam, than when the plain metal is used. This disadvantage is partly to be overcome by frequent cleaning-up. When ores contain minerals injurious to the mercury-surface, Muntz metal is preferable.

Sickening is prevented in the presence of base metals by the action no doubt of the zinc, forming one of the components of the alloy, which liberates hydrogen, and so exerts a powerful reducing effect. Muntz metal plates are easier therefore to keep in order than copper. They are hardly ever affected by sickening, and verdigris (caused by the presence of impurities in the ore and battery water) is remarkable for its absence.

Formerly at the Saxon mill, a 7 lbs. jar of potassium cyanide, costing 23s., was needed monthly for dressing the plates. After introducing Muntz metal, half a bottle of sulphuric acid (costing 3s. 4d.) sufficed for the same time. With highly acid ores (heaps of waste and old mullock-tips), copper, however, is to be preferred, as a scum is formed on the Muntz metal, while the free acid in the stone tends to keep the copper clean. In both cases it must be understood, however, that the amalgamation is interfered with.

At the Cambria, a customs mill, both varieties of plates are used: Muntz metal for the top, and copper for the bottom of the table, so as to meet the requirements of different kinds of ore. The result of experience in the Thames mills has been therefore to recommend Muntz metal for amalgamating plates where poor ore is being crushed, also in custom

milling, and where the ore is charged with minerals which injuriously affect the mercury. For material containing acid waters or very rich ore, unaccompanied by a large proportion of sulphides, copper-plates are preferable. In general, Muntz is the cheaper metal of the two, it lasts longer, facilitates a rapid and easy clean-up, and requires less attention. These points should recommend it for customs mills.

In dressing new Muntz metal plates the following are the steps to be taken :—Rub the surface of the plate with fine clean sand to get it mechanically clean, then wash it with a weak (1 to 6) solution of sulphuric acid to make it chemically clean. Then start to rub in a little mercury; rubbing in one place till it bites, that is a spot begins to amalgamate. Give a circular movement to the flannel or mop ; once started, the amalgamation spreads in ever-widening circles.

There is something generally to be learnt from acquaintance with each new mill and camp, each district has a lesson to give and every mill offers some suggestion.

Practice at Clunes, Victoria.

Clunes is rendered famous as the locality where Mr. J. W. Esmond discovered the first gold found in the colony, on June 29th, 1851 ; though not equal in importance as a mining centre to either Ballarat or Bendigo, it has done most useful work in the development of the mining and milling practices of the Colonies.

The history of the old Port Phillip batteries of the Port Phillip and Colonial Company, has had an important effect on colonial quartz reefing ; a model which has left its impress on the system in vogue at the present day. At the date when crushing first commenced in May, 1857, the treatment of gold-quartz was a problem entirely unsolved, and the Port Phillip mill laid down the basis of modern Colonial ideas on the subject. While assays proved the loss in the tailings in 1861 to amount to 6 dwts. 1 grain per ton, by numerous changes (suggested by careful experiments) this loss was decreased, till in 1870, it had been gradually reduced to 17 grains per ton. It should be remarked, however, that the average grade of the ore appears to have also altered, the yield averaging 12 dwts. 20 grains in 1861, as against 5 dwts. 17 grains in 1870.

In 1864, the plant was increased to 80 heads, and the first buddles were placed in position in 1865. The first rock-breaker was then introduced. The mine paid its highest dividends (£48,271 17s. 6d.) in 1867, on ore valued at 8 dwts. 23 grains per ton, with a loss of 2 dwts. 7 grains in the tailings. While from 1857 to 1881, it produced 1,204,908 tons of quartz, which yielded gold to the value of £1,946,989, out of which £481,455 were distributed in dividends.

The ore from the mine passes through the rock-breakers, preceded by sizing-bars (grizzleys), before entering the mill, which was built in different sections, at various periods.

No. of Heads.	Weight of Heads or Shoes. Cwts.	Date of Erection.
20	$2\frac{1}{2}$	1857
24	$2\frac{1}{2}$	1858–1859
12	$2\frac{1}{2}$	1860
24	$3\frac{3}{4}$	1864

There are four stamps to each box. The stamp-heads and shoes are square. The mortars have a back-and-front discharge. The daily crushing capacity is at the rate of 2 tons 12 cwts. for the light stamps, and 3 tons 12 cwts. for the heavier ones. The speed is 82 drops per minute, and the height of drop is 8 inches. The issue or depth of discharge is maintained as far as possible at $4\frac{1}{2}$ inches. The grating is of copper, pierced with 81 round holes per square inch.

The pyrites concentrated on the Munday buddles have amounted to $\frac{3}{4}$ per cent. of the ore crushed. Its average contents have been 4 ounces 1 dwt. 14 grains per ton. The bullion is 965 fine. The retorting percentage has averaged 38.

The business of the mill has always been carried out in a most systematic manner. The following statement of product is from the mill records for the four weeks ending May 21st, 1873:—

Where Amalgam was Produced.	Oz.	Dwts.	Retorted. Ozs.	Dwts.	Per Cent. of Total.
Mortar-box (bed)	1,466	0	673	11	59·05
Wells	708	3	249	12	21·87
Blankets	408	2	121	12	10·66
Mills	383	0	96	7	8·45
Total	2,965	5	1,141	2	

Other statistics kept were as follows:—Number of stamps, 80; tons crushed, 5,023; hours worked, 518 or 21·58 days. Average duty per stamp, 2·9 tons; yield per ton, 4 dwts. 10·12 grains; loss in tailings per ton, 20·16 grains; contents per ton, 5 dwts. 6·28 grains.

The amalgam coming from the mortar-box retorts 46 per cent.; from the wells, 35 per cent.; from the blankets, 30 per cent.; from the Chilian mill, 25 per cent. Of the total product obtained by direct amalgamation more than half came from the mortar-box, indicating the free-milling character of the ore. Of the total, 80 per cent. went no farther than the wells immediately outside the box. The above yield, deducting loss, represents an extraction of 84 per cent.

The batteries of the Port Phillip works are now idle, but the milling practice they inaugurated is seen reproduced in the newer mills of the South Clunes, and Dixon's North Clunes works.

The following table illustrates their different features. It may be mentioned that prior to 1865, the ore was calcined to render it more easily broken. This practice has not yet altogether died a well-merited death in Victoria and New South Wales:—

Name of Mill.	Port Phillip.	South Clunes United.	Dixon's North Clunes.
Number of stamps	56, 24	60	30
Weight of stamps (lbs.)	728, 896	896	896
Drops per minute	82	80	80
Height of drop (inches)	8	8	8
Depth of discharge (inches)... ...	4½	4½	7
Capacity per stamp (tons)	3	2½	3½
Total capacity (tons)...	240	150	100
Grating		Copper-plate.	
Holes per square inch	81	100	180
Concentrates (per cent.)	¾	⅝	3
Contents of concentrates	4 ozs. 1 dwt.	3 ozs. 5 dwts.	3 ozs.
Bullion fineness (per 1,000)	970	968	978
Loss of mercury per ton (grains) ...	5¾	5½	5½
Wear of grating (days)	30	25	?
Water per stamp per minute (gallons)	6	8	10
Retort (per cent.)	38	42	40

At the South Clunes United mill, as the die wears down, sand is packed underneath, and when about 2 inches have worn away, a sectional false-bottom is placed underneath; in this way the depth of discharge is kept fairly constant at 4½ inches.* Each section of the false-bottom consists of a plain iron casting of sufficient length to serve for two dies, the centre die being supported by a section of half the length of the others. The rate of crushing averages 2·4 tons per 24 hours.

The screen is of copper-plate, weighing 1½ lbs. per square foot, and is perforated with 100 holes per square inch. The average wear is about 25 days, working full time. Iron punched gratings scarcely lasted a week. The concentrates have increased with the depth of the workings from ¾ per cent. to 1 per cent. The concentrates usually carry 3 ounces of gold. Lately, however, they have become poorer, 28,820 tons of ore yielding 178 tons 19 cwts. 3 qrs. of pyrites worth £560 12s. 8d. The percentage of gold in the amalgam varies from 36 to 45.

The mill-stuff is discharged into ore-bins, but the ore is fed by self-

* In California, wooden blocks, set so as to raise the screen framing, serve the same purpose.

feeders, of simple design, to the boxes, which have a front-and-back discharge.* The battery pulp passes through wells and then over blankets.

The blanket washings are treated in revolving barrels with mercury. The tailings go to Cornish buddles supplemented by ties outside the mill. Tramming the ore one-third of a mile to the mill and breaking it, costs 8d. per ton.

The feeder before-mentioned, is somewhat similar to an arrangement common in German stamp-batteries, consisting of a rod which has a round iron disc keyed to its upper end, projecting below a false or extra tappet attached to the middle stamp of a battery of five heads. The shoe (fixed to the lower end of this rod) gives a shock to the shoot leading from the ore-bin to the battery-pocket. Whenever the ore inside the box has worked down low enough for the false tappet to strike the disc of the rod, a fresh quantity of stone is thrown into the mortar.

The discs are keyed on. The order of drop of the heads is 5, 3, 4, 2, and 1. The shoes are of cast-iron, 10 inches in diameter and 10 inches high. The dies are hexagonal, of wrought-iron,† with a diameter of 10 inches and depth of 6 inches. New shoes weigh 196 lbs., new dies 140 lbs. A shoe will crush 90 tons and a die 420 tons before it is worn out Cast-iron shoes cost 12s. 6d. per cwt. and wrought-iron dies 11s. 6d. per cwt. delivered at the mill.

The gratings are vertical and covered by a splash-box which slopes forward. The front grating-frame is 5 feet by 13 inches, while the back one is 5 feet by 12 inches. The pulp discharged from the back, is led round to the front of the box, by a launder.

The whole of the battery discharge falls on to an iron plate in front of the lip of the mortar, which is $\frac{3}{16}$ inch thick, punched with $\frac{5}{16}$ inch holes, drilled at the four corners of a square inch, and is called the distributor. Its function is to spread the ore over the whole width of the tables and wells, and to catch any of the coarse stone in the battery-bottoms in case of a screen happening to break accidentally at the bottom, which is liable to occur with iron screens with a low discharge, but is generally an indication of over-feeding. In cases of this sort, where amalgamated plates are used under the lip of the box, it may save amalgam from being scraped off and lost, and as pointed out elsewhere it has a tendency to form

* The use of a double-discharge is limited to ores which cannot be amalgamated in the battery with advantage. This is a case in point, another one is with a high sulphide ore, especially if it contains brittle sulphides, which from over stamping are liable to be slimed.

† In America, steel shoes and iron dies are used in some mills, it being found that the iron die wears more evenly than a steel one would do.

thick rough bosses and ridges of amalgam, for a width of several inches, where the streams of water fall on the copper-plates.

At the South Clunes mill, the pulp from the distributor is spread over a plain wooden apron 20 inches wide and 2 inches thick which further aids its distribution. Two wells succeed this, which are guarded from theft by a wooden grating kept under padlock. The first well has a drop of 10 inches and a depth of 4 inches. It holds 50 lbs. of mercury, and the pulp in passing through it, is forced under a narrow upright board (running lengthways along the well close to its edge, on the side nearest the battery) in contact with the quicksilver. The second well, which follows immediately after, has a drop of 8 inches, is 4 inches deep, and also holds 50 lbs. of mercury. These wells, including the lip, are of cast-iron, and have a curved inside contour. They are sunk into the wood of the frame holding them. Iron wells appear to have a tendency to keep the mercury quick and lively. At Clunes they have an inside diameter of 3 inches, and are placed so as to have a slight slope to one end, where a tap-hole facilitates the removal of the mercury at cleaning-up time.

The pulp passes from the wells to the blanket tables, which have a width that takes in two batteries of five heads each. This width is sub-divided by seven partitions, 18 inches wide and 12 feet long. The grade is $\frac{3}{4}$ inch per foot. Then follow five circular improved Cornish buddles (Munday), and finally the tailings pass over ties outside the building. These last have a length of 20 feet and a fall of 1 inch per foot. The gold saving is effected by the mortar-box and wells, and indirectly by the blankets, buddles, and ties. No mercury is employed in the box, which the use of copper-gratings would in itself prevent, while the very free character of the gold, does not necessitate its use at this stage of the treatment. The mortar-box is roomy, and gives the gold an opportunity to separate from the pulp by the action of gravity alone. The interior length is 58 inches, interior width 16 inches, width between dies 1 inch, with an extra inch between the side dies and the ends of the box. Distance from dies to back of mortar 4 inches, from dies to screen 3 inches, and between centres of dies 11 inches.

In cleaning-up, the gratings are removed, and the material inside and around the dies is shovelled into buckets and passed over a strong wire riddle* 2 feet in diameter and No. 4 mesh. One of these lasts a year. The roughs from this operation go back to the mortar-box to reset the dies before restarting, $1\frac{1}{2}$ to 2 buckets being obtained at each fort-

* In Queensland, they are generally run through a tom or rocker, and the pyrites caught in the riffles is ground in pans.

nightly clean-up. The fines are sifted into a blanket-tub, and then introduced into an amalgamating-barrel. There are five such barrels in the mill, with a capacity of 54 gallons each. They make 16 revolutions per minute and are worked from 8 to 12 hours, 10 hours being the average time. The water is used cold; 75 lbs. of mercury are added to each charge with a bucket full of wood-ashes. When the amalgamation is finished, the contents of the barrel are discharged into a wooden tank, and passing through a perforated plate, flow over three drop-wells, with drops of 12 inches, 9 inches, and 6 inches respectively. Most of the amalgam is caught in the top well, the third is merely a safeguard. This disposes of the treatment of the residues.

The battery-wells are cleaned up once a week and the amalgam squeezed. The skimmings taken from the surface of the wells (largely consisting of heavy pyrites) are ground and amalgamated in three 3 feet berdans. The blankets are washed in tubs, the first row every hour, the second every alternate hour, and the third every third hour. With rich ore the washing is done more frequently. The blanketings caught, are treated in amalgamating barrels* in much the same way as the mortar-box residues. The material collected by the ties (straight troughs, in which heavy pyrites, etc., is settled by gravity), are also treated in the barrel. The tailings from all the barrels go to buddles. The concentrates thus obtained are roasted in a reverberatory furnace, then ground in a Chilian mill with the addition of mercury, by which the gold is amalgamated. The total yield of the mill in one month, crushing 2,973 tons of stone, which yielded 981 ounces 19 dwts. 12 grains of gold was thus distributed:—

	Amalgam. Ozs.	Dwts.		Bar-gold. Ozs.	Dwts.		Retort Percentage.
Mortars	955	5					
Wells	644	19	...	840	19	...	36 to 48
Blankets (by the barrels) ...	364	15					
Skimmings (by the berdans) ...	167	14	...	53	2	...	32
Tailings (by the ties and barrels)	24	10	...	7	5	...	30
Concentrates (pyrites, 17 tons)	310	4	...	80	13	...	26

Neglecting the concentrates and tailings, of the total amalgam caught, the percentage is therefore thus distributed:—Mortars, 44·8 per cent.; wells, 30·2 per cent.; blankets, 17·1 per cent.; skimmings, 7·9 per cent. The loss of mercury for the past seven years has averaged 5½ grains per ton of ore crushed. Occasionally the loss has risen to 1½ ounces per ton of ore, due to formation of copper-amalgam, which like lead-amalgam floats on

* In California, an Attwood amalgamator sometimes serves for the amalgamation of the blanketings.

the mercury, and is readily carried off with the tailings. The presence of the copper cannot be attributed to the copper gratings, but to particles of native copper in the ore. At one time, 80 ounces of copper were collected in one month from the skimmings of the wells. The total consumption of mercury (inclusive of treatment of pyrites in Chilian mills) during eleven months from July 1st, 1891, to June 30th, 1892, amounted to 3,302 lbs., when crushing 309,400 tons of ore, equivalent to a loss of 3 dwts. per ton of ore.

Up to the year 1879, according to information furnished by Mr. Hewitson, the manager, the gratings used were imported from England, being made of copper $\frac{1}{16}$ inch thick, drilled with 81 holes per square inch. When in full work the imported grating lasted twelve months, 2,200 tons of ore passing through each grating. At the Port Phillip mill, owing to its smaller area, the life of a grating reached $1\frac{1}{2}$ years. The protective tariff caused the imported grating to become too expensive, and one of domestic manufacture took its place. This wore for less than half the time, but six times as long as ordinary punched iron. The experience with the present lighter type of copper grating has been very good. During the last seven years, 258 gratings have been used up. Their cost was £197 6s., and during that time 181,792 tons were crushed, or at the rate of 355 (long) tons during the life of a grating. It was found that ordinary punched Russian iron lasted only one quarter the time of the new style of grating. Baize is used for the blanket strakes; the cost of this item in one year was £49 7s. 2d. During the same period the wages at the mill amounted to £1,306 4s. 9d., treating 28,820 tons of ore, or $10\frac{1}{2}$d. per ton.

The total cost of milling, including supplies, wear and tear, treatment of pyrites, etc., amounted to 2s. 3d. At Dixon's North Clunes mill the front grating is copper with 180 holes per square inch, while the back grating is brass wire with 230 holes to 240 holes per square inch. They have 6 patent Munday buddles with iron scrapers, 2 to each 10 stamps. The pyrites is washed and treated in a Chilian mill at a cost of £1 16s. 11d. per ton. The roasting of 85 tons 1 cwt. cost £89 6s. 7d., and grinding £67 15s. 11d., or a total cost of £157 2s. 6d.

The smaller capacity of South Clunes United mill, as compared with the other two referred to in the table, may be accounted for by the absence of a rock-breaker.

The finer gratings of the Dixon's North Clunes mill are offset by the less depth of discharge of the other batteries. The retort percentage of the North Clunes mill, notwithstanding that the gold in the ore is of a coarser

nature, is not quite so high as that of the South Clune's, owing no doubt to the finer size of grating. The large quantity of water used in the mills of this district is accounted for by the double-discharge of the mortars and the use of very wide blanket-tables.

The loss of mercury is exceedingly low: 5½ grains of mercury per ton being probably one of the smallest on record,* this may be attributed to the avoidance of the chief cause of loss in an ordinary mill, viz. : flowering in the battery, which cannot of course happen here.

The ore treated in these mills is broken from quartz-veins, traversing slate and sandstone-beds. When sent to the mill, the quartz is accompanied by a small admixture of country rock. The quartz is white, often honeycombed, and sometimes saccharoidal. The gold is coarse, often of very high caratage, frequently visible to the naked eye, and arranged for the most part along the faces of small fractures, or seams in the quartz; a blow therefore tends readily to detach it. Occasionally the percentage of mullock (waste rock) increases considerably, and the gold is accompanied by pyrites, chiefly composed of arsenic and iron sulphides, or occurs in a matrix consisting of slate and quartz intermixed.

The Port Phillip and Colonial mill-book shows that the proportion of the total yield of gold coming from the mortars and wells steadily declined from 1868 (when it stood at 87·03 per cent.) to 1879 (when it stood at 68·49 per cent.). On the other hand, however, the yield from the blankets and concentrates increased correspondingly. The explanation lies in the fact that as the mine workings reached a greater depth, the ore, by the steady increase of the pyrites it contained, became less free-milling.

This reasoning is confirmed by the returns, for while in 1866, the yield of concentrates amounted to 268 tons, averaging 2 ounces 19 dwts. 4 grains from the crushing of 59,578 tons; in 1879, the pyrites amounted to 421 tons, averaging 4 ounces 15 dwts. 20 grains, resulting from the treatment of 56,766 tons. In 1870 the use of blankets was discontinued, but in 1873 it was resumed; in the interval the yield from the Chilian mill increasd considerably. In 1865, the ore yielded 7 dwts. 13¾ grains per ton; of which amount the boxes yielded 63·60 per cent.; the wells, 22·09 per cent.; blankets, 10·55 per cent.; mills, 3·76 per cent.; and mills and blankets, 14·31 per cent. In 1872, the ore yielded 4 dwts. 17¾ grains per ton; of which amount the boxes yielded 64·48 per cent.; the wells, 21·60 per cent.; the blankets, 1·06 per cent.; the mills, 12·86

* The record of greatest waste, it is stated, occurred at a mill in the Thames district, where 1 ton of mercury was used up in two weeks by a mill of 20 heads.

per cent.; and the mills and blankets, 13·92 per cent. In 1879, the ore yielded 8 dwts. 19¾ grains per ton; of which amount the boxes yielded 57·99 per cent.; the wells, 10·50 per cent.; blankets, 12·84 per cent.; mills, 18·67 per cent.; and mills and blankets, 31·51 per cent.

To consider the method in practice, it will be admitted that the use of costly chemicals in milling is as far as possible to be avoided. Mercury is generally a large item, and since 55 to 65 per cent. of the gold can be arrested in the box, without its use; the practice of the district in dispensing with it in the box, appears correct and advantageous. Under the stamps, it is liable to be flowered, *i.e.*, broken up into minute globules, which, collecting impurities, become covered with a film which makes them refuse to coalesce, causing them to be carried away on the surface of the water. With this loss of mercury there must also be a loss of gold, particles of which have entered into amalgamation with the escaping globules.

The absence of amalgamating-plates is remarkable, but in view of the character of the ore, Mr. Rickard considers it correct. Wells, as he states, are excellent-gold savers for ore of this type, in which the metal is both free and coarse. He points out as their advantages that they require less attention, their first cost is less than that of plates, and they are less affected by the occasional presence of minerals in the ore, injurious to amalgamation. Blankets, when intelligently used, are among the best of the simple contrivances known to mill-men, given as they are at Clunes plenty of width. At South Clunes there is a clear blanket space of 10½ feet. Ordinarily the slope of the blanket-tables would be 1¼ inches to 1½ inches per foot; but at Clunes, owing to the use of plenty of water due to the double-discharge, they have a pitch of only ¾ inch per foot. This is in itself an important factor, apt to be overlooked.

The after-treatment in the barrels may seem crude, but practice has shown it to be effective. The bad custom of putting pieces of iron into the barrel, with the idea of mixing and grinding-up the pulp, is here avoided, saving a large loss in flowered mercury. The means taken to keep the discharging depth of the mortar constant, is a factor of importance in the method pursued, which does credit to the practice of the district. The self-feeders do their work well, and though not perfect, are a great improvement on the bad and irregular hand-feeding which prevails in the majority of Colonial mills. The concentrating machinery may with reason be considered somewhat out of date, but the modified Cornish buddles in use, are doing excellent work. The use of the double-discharge mortar increases the crushing capacity of the mills, but also requires a much increased supply of water.

Speaking generally, the treatment the ore undergoes is remarkable, most of all on account of its simplicity, but so is the ore; and in this way the local practice carries out the first postulate of intelligent milling, viz., that the treatment should be varied according to the character of the ore to be dealt with. After a careful examination of the ore mined at Clunes, and of the milling it is subjected to, it is possible only to speak in words of commendation. To a mill-man it is almost solitary among the gold-mining districts of the Colonies, in being a quartz-milling centre which does not leave a feeling of dissatisfaction, and an impression of dissapointment on the visitor. You may visit mills in the most distant parts of Australia, and almost without exception, whenever you find good intelligent milling (and that does not happen often enough to be monotonous), the knowledge and experience of the individual in charge, have been obtained at Clunes.

The Port Phillip mill was the first to introduce the system of taking daily assays as a check on the work done in the mill.* In this respect the Clunes mill is still unfortunately a striking exception. In another department this mill was also a solitary pioneer, the first rock-breaker being introduced by the Port Phillip in 1865. Yet in Victoria to-day, there are only twelve rock-breakers!

Mr. Rickard expresses the belief that the work done by the Port Phillip and Colonial Company's mill, has been of more wide-reaching usefulness and more permanent benefit to the mining industry of Australia and New Zealand, than that of any other company which has gone into operation since the days of the discovery of gold, and records his conviction of the debt which quartz-milling in the colonies owes to its manager, Mr. R. H. Bland, who started its operations in 1856, conducted the numerous and valuable experiments which did so much to establish the correct basis of milling practice, and to-day still assists the industry by his sterling good sense.

* Mr. Rickard mentions as another instance, the mill of the Harrietville Gold Mining Co., Ld.; and it was also done at the Disraeli mill in Queensland, when under the writer's charge.

(*To be continued.*)

The following paper on "Mining Explosives: Their Definition as Authorized under the Explosives Act, 1875," by Mr. A. C. Kayll, was taken as read:—

MINING EXPLOSIVES: THEIR DEFINITION AS AUTHORIZED UNDER THE EXPLOSIVES ACT (1875).

By A. C. KAYLL.

The Explosives Act, 1875 (38 Vict., c. 17), came into operation on January 1st, 1876. Briefly stated, the Act requires that all factories, stores, and explosives shall be licensed, and consequently these come under Government supervision.

The term explosive is defined as follows:—

"1. Means gunpowder, nitroglycerine, dynamite, gun-cotton, blasting-powders, fulminate of mercury or other metals, coloured fires and every other substance, whether similar to those above-mentioned or not, used or manufactured with a view to produce a practical effect by explosive or a pyrotechnic effect; and

"2. Includes fog-signals, fireworks, fuzes, rockets, percussion-caps, detonators, cartridges, ammunition of all descriptions, and every adaptation or preparation of an explosive as above defined."

Explosives are classed by Orders in Council as follows:—

Class 1. Gunpowder.
„ 2. Nitrate mixtures.
„ 3. Nitro-compounds:
Division 1.
Division 2.
„ 4. Chlorate mixtures.
„ 5. Fulminate.
Class 6. Ammunition:
Division 1.
Division 2.
Division 3.
„ 7. Fireworks:
Division 1.
Division 2.

Class 1.—Gunpowder is not included in this paper as its component parts are so well known.

Class 2.—Nitrate mixtures, other than gunpowder, such as safety blasting-powder, fortis explosive, etc.

Class 3, Division 1.—Nitro-compounds, containing nitroglycerine, such as dynamite, gelignite, stonite, ardeer powder, carbonite, etc.

Division 2.—Nitro-compounds, *not* containing nitroglycerine, such as gun-cotton, tonite, ammonite, roburite, schultze gunpowder, etc.

Class 4.—Chlorate mixtures, containing (especially) chlorate of potassium, such as asphaline. All chlorate mixtures are liable to spontaneous ignition, and are extremely sensitive to friction and percussion, hence, with the exception of asphaline, no chlorate mixture has been authorized in this country.

It is interesting to follow the history of dynamite during the past few years. In 1876, dynamites Nos. 1 and 2 were the only nitroglycerine compounds in use for mining purposes; No. 2 is little used in this country, and its definition remains the same up to the present time.

Up to 1886, other varieties were added, such as E.C. dynamite, dynamite 1 S., dynamite 1 S.B., but in that year the Nobel Dynamite Trust Co., Limited, acquired the shares of certain explosive companies, which led to a new and more comprehensive definition of dynamite No. 1, embracing the varieties known as 1 S., E.C., and No. 1 S.B.

In 1876, a new and important nitroglycerine and nitro-cotton compound was licensed under the name of blasting-gelatine, which in 1880, extended to four varieties, viz., blasting-gelatine Nos. 1, 2, 3, and 4. In 1883, the varieties known as Nos. 3 and 4 were removed from the list, and reappeared in the following year under the name of gelatine-dynamite Nos. 1 and 2. In 1886, another variety of this last No. 2 appeared under the name of gelignite.

These gelatinized compounds have almost entirely superseded the use of dynamite. The advantages of these compounds in mining are many: their being unaffected by water, their plastic nature, and their being more cleanly to use, are among the most important. On the other hand, these explosives are liable to exude a thin gelatine, which is liquid at ordinary temperatures, and when frozen they are very sensitive to a blow; the reverse holds good with dynamite.

With regard to the new class of explosives containing other nitro-compounds mixed with nitrates, such as ammonite, bellite, roburite, securite, etc., their less sensitiveness to percussion and friction while giving them the advantage of greater safety, exposes them to miss-fires, which become more frequent as the compounds age and absorb moisture.

With reference to the addition of ammonium salts to nitro-compounds, Dr. Dupré says in his report of 1890 that, while the addition of ammonium salts to dinitro-benzole has been sanctioned, the addition of ammonium salts, other than the carbonate, to explosives containing gun-cotton or nitroglycerine, has always been reported against. The assigned reason being that all ammonium salts, especially when exposed alternately to

moist and dry air at slightly elevated temperatures, lose traces of ammonia and become acid. Now, nitro-compounds, like dinitro-benzole, are little, if at all, affected by traces of acid, and under such circumstances show no tendency to spontaneous decomposition which might lead to ignition or explosion. Nitro-compounds, like gun-cotton and nitroglycerine (more strictly speaking nitric ethers), on the other hand, are seriously affected by traces even of acids, especially strong mineral acids, and decomposition once started goes on and ultimately leads to total decomposition, which may end in ignition or explosion. Hence ammonium salts exert no dangerous action on true nitro-compounds, but may fatally affect the stability of nitric ethers like gun-cotton and nitroglycerine.

It should be clearly understood that all the explosives mentioned in the following list are not necessarily manufactured, and, as a matter of fact, many of them exist only on paper.

Importation.

The amount of dynamite imported prior to 1881 was inconsiderable. (The import trade became established in 1881 after the expiration of Nobel's English dynamite patents in March of that year).

The following table shows the quantities imported* since that time :—

	Dynamite. Lbs.	Gelatinous forms. Lbs.
1882 ...	1,008,050 ...	—
1883 ...	1,920,650 ...	—
1884 ...	1,085,000 ...	—
1885 ...	1,068,100 ...	—
1886 ...	1,073,800 ...	20,000
1887 ...	707,500 ...	No return.
1888 ...	1,120,800 ...	„
1889 ...	1,155,200 ...	154,250
1890 ...	371,650 ...	462,002
1891 ...	185,200 ...	478,650
1892 ...	226,398 ...	492,270

Dynamite Nos. 1 and 2, E.C. Dynamite, Dynamite 1 S., Dynamite No. 1 S.B., and Dynamite No. 0.

In 1876, on the passing of the Explosives Act, dynamites Nos. 1 and 2 only were licensed, the following being the authorized composition under Class 3, Division 1 :—

Dynamite No. 1 consisting of not more than 75 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 25 parts by weight of an infusorial earth, known as "kieselguhr," and sufficiently absorbent in quality when mixed in the above proportions to prevent exudation of nitroglycerine.

* The amount imported is not necessarily for home consumption, a considerable quantity being transhipped to other countries.

Dynamite No. 2 consisting of not more than 18 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 82 parts by weight of a pulverized preparation composed of nitrate of potash 71 parts, charcoal not less than 10 parts, and purified paraffin (or ozokerit) 1 part (or nitrate of potash 72 parts and charcoal not less than 10 parts) by weight, and sufficiently absorbent in quality when mixed in the above proportions to prevent exudation of nitroglycerine.

The above definition of dynamite No. 2 is extant at the present time (1892).

No change was made in this explosive until 1887, but in 1883, E.C. dynamite and in 1884, dynamite 1 S. were added to the list. Both of these are defined in identical terms, viz. :—

E.C. Dynamite and Dynamite 1 S consisting of not more than 75 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 25 parts by weight of a preparation consisting of an infusorial earth known as kieselguhr and carbonate of soda, the said preparation being sufficiently absorbent in quality, when employed in the above proportions, to prevent exudation of nitroglycerine. Provided that the amount of carbonate of soda present shall not exceed 3 parts by weight in every 100 parts by weight of the finished dynamite.

In 1886, a company under the name of the Nobel Dynamite Trust Company, Limited, was formed, having for its primary object the acquisition by exchange of shares of certain explosive companies, and this led to a change in the list in the following year.

In 1887, dynamite No. 1 S.B. was licensed temporarily, and as the authorized composition is covered by the new definition of dynamite No. 1 (see below), it is unnecessary to give it here.

A new and more comprehensive definition of dynamite No. 1 was substituted for the former definition. It included the varities known as dynamite No. 1, dynamite 1 S., E.C. dynamite, and the new dynamite No. 1 S.B. Consequently, these four were removed from the (1887) list of explosives, and the authorized compositions became :—

Dynamite No. 1 consisting of not more than 75 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 25 parts by weight of :—(*a*) an infusorial earth known as kieselguhr, or (*b*) a non-explosive mixture of kieselguhr with such other ingredients and in such proportions as may for the time being be sanctioned by the Secretary of State, provided (1) that the said (*a*) kieselguhr or (*b*) non-explosive mixture shall be sufficiently absorbent in quality when mixed in the above proportions to prevent exudation of nitroglycerine ; and (2) that there may be added to the kieselguhr, or non-explosive mixture, an amount of carbonate of ammonium not exceeding 1½ parts by weight in every 100 parts by weight of the finished dynamite.

The Secretary of State has signified his approval of the use of one or more of the following ingredients in admixture with kieselguhr in dynamite No. 1, viz. :—Carbonate of sodium, sulphate of barium, mica, talc, or ochre, 8 parts or (less) by weight in substitution for an equal

amount by weight of kieselguhr. Provided that the total amount of carbonate of sodium present shall in no case exceed 3 parts by weight in every 100 parts by weight of the finished dynamite.

Dr. Dupré in his chemical report for the (1887) year describes the alteration as follows :—The addition of sodium carbonate, barium sulphate, mica, talc, and ochre to the ingredients composing dynamite No. 1 has been authorized subject to the following conditions :—(*a*) That the total amount of these substances shall not exceed 8 per cent. of the finished dynamite. (*b*) That the amount of sodium carbonate shall not exceed 3 per cent. of the finished dynamite.

These additions were asked for mainly in order to reduce the too highly absorbent quality of some kinds of kieselguhr, and thus enable the manufacturer to produce a 75 per cent. dynamite with the absorbing capacities of the infusorial earth nevertheless fully saturated.

In 1890, the addition of sulphate of magnesium to dynamite was approved, but this entailed no alteration in the definition, the object of the addition being to render the dynamite flameless.

No alteration has been made since 1887, and the above are the authorized definitions of dynamite up to the end of 1892.

The makers define the composition of dynamite No. 1 as containing 75 parts of nitroglycerine and 25 parts of kieselguhr.

Dynamite No. 0, a new dynamite, was examined in 1892 and favourably reported on, but does not appear on the authorized list for that year. It consists of 74 parts of nitroglycerine absorbed in 26 parts of nitrocotton.

Blasting-gelatine and Camphorated Gelatine.

Blasting-gelatine came into existence in 1878, but was not manufactured for general sale previous to 1879. It was licensed under Class 3, Division 1, as follows :—

Blasting-gelatine No. 1. consisting of collodion cotton, as hereinafter defined, combined with thoroughly purified nitroglycerine, and containing not less than 7 and not more than 10 parts of such collodion cotton in every 100 parts, the whole to be of such a character and consistency as to prevent liquefaction or exudation. Collodion cotton, consisting of nitro-cellulose, thoroughly washed and purified, and of such composition that at least 95 per cent. of the dry material shall be soluble in a mixture of ether and alcohol.

In 1880, the above definition was altered, and three new kinds were added to the list. The following being the authorized definitions :—

Blasting-gelatine No. 1, consisting of nitro-cotton, as hereinafter defined, combined with thoroughly purified nitroglycerine, and containing not less than 7 parts of such nitro-cotton in every 100 parts of blasting-gelatine, the whole to be of such

character and consistency as not to be liable to liquefaction or exudation. Nitro-cotton, consisting of nitro-cellulose, carefully washed and purified, and of such composition that not less than 70 per cent. of the dry material shall be soluble in nitroglycerine.

Blasting-gelatine No. 2, consisting of blasting-gelatine No 1, as above defined, mixed or incorporated with nitrate of potash (with or without charcoal) or such other nitrate as may from time to time be sanctioned by a Secretary of State.

Blasting-gelatine No. 3, consisting of thickened nitroglycerine, as hereinafter defined, mixed or incorporated with one or more of the following non-explosive ingredients, viz.:—Cotton, charcoal, kieselguhr, or such other ingredients as may from time to time be sanctioned by a Secretary of State, the whole to be so mixed and to be of such a character as not to be liable to liquefaction or to separation of ingredients. Thickened nitroglycerine, consisting of thoroughly purified nitro-glycerine, combined with not less than 4 parts of nitro-cotton, as above defined, in every 100 parts of thickened nitroglycerine.

Blasting-gelatine No. 4, consisting of blasting-gelatine No. 3, as above defined, mixed or incorporated with nitrate of potash, or such other nitrate as may from time to time be sanctioned by a Secretary of State.

There seem to have been great difficulties in the manufacture of this explosive, and in 1882 its manufacture was suspended. Dr. Dupré in his report gives exudation of nitroglycerine as the cause of every rejection: this led in 1884 to an alteration in the definitions of blasting-gelatine Nos. 1 and 2, and blasting-gelatine Nos. 3 and 4 were removed from the list, reappearing under the designations of gelatine-dynamite Nos. 1 and 2.

In 1884, the authorized definitions were as follows:—

Blasting-gelatine No. 1, consisting of nitro-cotton, as hereinafter defined, combined with thoroughly purified nitroglycerine in such proportions that the whole shall be of such character and consistency as not to be liable to liquefaction or exudation. Nitro-cotton consisting of nitro-cellulose carefully washed and purified.

Blasting-gelatine No. 2, consisting of blasting-gelatine No. 1, as above defined, mixed or incorporated with nitrate of potash (with or without charcoal) or such other nitrate as may for the time being be sanctioned by a Secretary of State.

Blasting-gelatine Nos. 3 and 4 appeared in 1884, under the designation of gelatine-dynamite Nos. 1 and 2.

In 1889, camphor was added to blasting-gelatine No. 1, and the mixture appeared on the list as under:—

Camphorated Gelatine, consisting of blasting-gelatine No. 1, as above defined, mixed or incorporated with camphor.

In 1890, the explosive was again adversely reported upon owing to exudation, which led to a slight alteration in the manufacture, appearing on the authorized list of 1891 under the following definition, viz.:—

Blasting-gelatine No. 1, consisting of nitro-cellulose, carefully washed and purified, combined with thoroughly purified nitroglycerine in such proportions that

the whole shall be of such character and consistency as not to be liable to liquefaction or exudation, and with or without carbonate of calcium or carbonate of magnesium not exceeding 1 part by weight in every 100 parts by weight of the finished explosive.

Blasting-gelatine No. 2. (Same as No. 2 above.)

Camphorated Gelatine, consisting of blasting-gelatine No. 1, as above defined, mixed or incorporated with camphor.

In 1890, the composition of blasting gelatine No. 1 was :—93 to 95 parts of nitroglycerine and 7 to 5 parts of nitro-cotton.

The composition of blasting gelatine No. 1 is stated by the makers in 1893 as :—92 parts of nitroglycerine and 8 parts of nitro-cotton.

There is no change in the definition of the above explosives for the present year (1892), but the samples examined show that the difficulties of exudation and liquefaction have not altogether been overcome.

Gelatine-dynamite.

In 1884, blasting-gelatine Nos. 3 and 4 disappeared from the list and reappeared under the designation of gelatine-dynamite Nos. 1 and 2. In appearance this explosive resembles blasting-gelatine, and occupies a position midway between dynamite and blasting-gelatine. The authorized definitions under Class 3, Division 1, were as follows, viz. :—

Gelatine-dynamite No. 1, consisting of thoroughly purified nitroglycerine, thickened by being combined with nitro-cotton, as hereinafter defined, and mixed or incorporated with one or more of the following non-explosive ingredients, viz. :—Cotton, charcoal, or such other ingredients as may for the time being be sanctioned by a Secretary of State, and in such proportions that the whole shall be of such character and consistency as not to be liable to liquefaction or exudation. Nitro-cotton, consisting of nitro-cellulose carefully washed and purified.

Gelatine-dynamite No. 2, consisting of gelatine-dynamite No. 1, as above defined, mixed or incorporated with nitrate of potash, or such other nitrate as may for the time being be sanctioned by a Secretary of State.

This definition, with slight alteration in the wording, is extant at the present time (substitute nitro-cellulose for nitro-cotton and omit definition of nitro-cotton.)

In 1885, wood-meal, for the bleaching or purifying of which no chemical agents have been used, or if used have been absolutely removed, was authorized as an ingredient in this explosive.

In 1889, the chemical report upon gelatine-dynamite was far from satisfactory owing to exudation, but a more satisfactory report appeared in 1890.

No changes having been made since 1890, the definition of gelatine-dynamite is as above (1884) with a slight alteration of the wording.

The makers define gelatine-dynamite as consisting of :—80 parts of a mixture composed of 93 parts of nitroglycerine and 7 parts of nitro-cotton, and 20 parts of a mixture consisting of 80 parts of nitrate of potassium and 20 parts of wood-meal.

Gelignite.

In 1886, varieties of gelatine-dynamite No. 2, made with 60 to 65 per cent. of thin blasting-gelatine, were issued under the name of gelignite.

The name of gelatine-dynamite No. 2, under which they were originally licensed is, however, still retained upon the boxes, as the official name of the explosive in addition to that of gelignite.

In 1888, the admixture with gelatine-dynamite Nos. 1 and 2 of one or other of the following ingredients to an extent not exceeding 1 per cent. was authorized, viz., carbonate of lime or carbonate of magnesia.

The makers define gelignite as consisting of :—65 parts of a mixture composed of 94 parts of nitroglycerine and 6 parts of nitro-cotton, and 35 parts of a mixture composed of 80 parts of nitrate of potassium and 20 parts of wood-meal.

Gun-cotton.

Gun-cotton is defined at the present time, under Class 3, Division 2, as—

Gun-cotton, consisting of nitro-cellulose thoroughly purified, with or without the addition of carbonate of calcium.

Gun-cotton was first discovered in 1832, under the name of xyloïdine, but no practical result was obtained until 1845. It is largely used for war purposes, the gelatinized forms of nitroglycerine have to a great extent superceded it for mining purposes, its rigidness and bulk precluding its general use in bore-holes.

Saturated gun-cotton is uninflammable and requires an abnormal amount of fulminate to detonate it, but it is easily exploded by the initial detonation of the dry material in contact with it. Gun-cotton does not freeze, neither does any liquid explosive exude from it.

Lithofracteur.

This is somewhat similar to dynamite No. 2, and was one of the six nitroglycerine preparations for which licenses were granted up to December 31st, 1875, and from that time up to the present, no alteration has been made in the authorized definition, which stands as follows under Class 3, Division 1 :—

Lithofracteur, consisting of not more than 55 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 45 parts by weight of a pulverized preparation consisting of 1 part by weight of charcoal, bran, and sawdust (or of any

one or more of the same), 3½ parts by weight of an infusorial earth known as kieselguhr, 2½ parts by weight of nitrate of baryta and bicarbonate of soda (or of either of them), ½ part by weight of sulphur and manganese (or either of them), and sufficiently absorbent in quality when mixed in the above proportions to prevent exudation of nitroglycerine.

Colonel Cundill records in 1889 analyses as below, and remarks that lithofracteur is inferior in explosive power to dynamite No. 1, and is said to be more sensitive to heat. To all intents it is a mixture of ordinary dynamite with a crude sort of gunpowder. It is now rarely used in this country.

	A.	B.
Nitroglycerine	52	70
Kieselguhr and sand	30	23
Powdered coal	12	2
Nitrate of sodium	4	—
Nitrate of barium	—	5
Sulphur	2	—

Tonite or Cotton-powder; Liverpool Cotton-powder; or Potentite.

Tonite or cotton-powder was in use at the commencement of the Act (1875), and in 1880 a particular variety of nitrated gun-cotton previously covered by the designation of cotton-powder or tonite was re-named Liverpool cotton-powder or potentite. It was specified under Class 3, Division 2, as :—

Tonite or Cotton-powder No. 1, consisting of gun-cotton thoroughly purified, mixed or impregnated with a nitrate or nitrates.

A variety was licensed in 1881, as follows :—

Tonite or Cotton-powder No. 2, consisting of gun-cotton thoroughly purified, mixed or impregnated with a nitrate or nitrates and charcoal.

Liverpool cotton-powder or potentite was changed in name in 1882, first to potentite cotton-powder and subsequently to potentite. It is defined in identical terms with tonite No. 1.

Tonite or cotton-powder Nos. 1 and 2 were designated in 1887 as tonite Nos. 1 and 2.

A further variety was authorized in 1889, as follows :—

Tonite or Cotton-powder No. 3, consisting of a mixture of thoroughly purified meta-dinitro-benzole, and thoroughly purified gun-cotton mixed or incorporated with one or more of the following ingredients, viz. :—Nitrate of potassium, nitrate of sodium, nitrate of barium, and chalk.

The above definitions stand at the present time.

In 1890, tonite consisted of equal parts of nitrate of barium and gun-cotton. Dr. P. P. Bedson analyzed a sample (1890) as follows :—

Nitrate of barium	48·21
Nitrate of potassium	1·77
Gun-cotton	49·21
Moisture	1·58
	100·77

Sawdust and Gun-cotton-powder.

This explosive was licensed in 1876 under Class 3, Division 2, and is authorized at the present time as :—

Sawdust and Gun-cotton Powder, consisting of a mixture of two or more of the following explosives, viz. : Sawdust gunpowder, gun-cotton, and cotton gunpowder.*

Pudrolithe or Rock-powder.

Pudrolithe was authorized in 1876, under continuing certificate in Class 2, and amended in 1878, the following being the authorized composition :—

Pudrolithe consisting of a mixture of any two or more of the following ingredients, viz. :—Tan, sawdust, nitrate of soda, nitrate of baryta, charcoal, sulphur, and saltpetre.

This explosive was withdrawn from the list in 1882.

The following composition is given by Colonel Cundill :—Saltpetre, 68 parts ; sulphur, 12 parts (or sulphur, 8 parts ; powdered gumlac, 4 parts) ; charcoal, 6 parts ; nitrate of barium, 3 parts ; nitrate of sodium, 3 parts ; sawdust, 5 parts ; and spent tan, 3 parts.

Schultze Gunpowder and Schultze Blasting-powder.

Schultze gunpowder was manufactured prior to the Explosives Act, 1875, and in 1876 Schultze blasting-powder was added to the list. The actual date of the invention of the explosive as patented by Mr. Schultze is 1868. The definition of these explosives was altered in 1884, and appeared under Class 3, Division 2, in the list as follows :—

Schultze Gunpowder, consisting of nitro-lignin, carefully purified and mixed or impregnated with a nitrate or nitrates other than nitrate of lead, and with or without starch or collodion (such collodion to consist of carefully purified nitro-lignin dissolved in commercially pure ether and alcohol) or pure solid paraffin, provided that such paraffin shall be free from mineral acid.

Schultze Blasting-powder, consisting of Schultze gunpowder, as above defined, mixed with charcoal or sugar.

In 1892, the definition was altered to the following :—

Schultze Gunpowder, consisting of nitro-lignin, carefully purified and mixed or impregnated with a nitrate or nitrates other than nitrate of lead or nitrate of ammonium, and with or without starch or collodion (such collodion to consist of carefully purified nitro-lignin dissolved in a safe and suitable solvent) or pure solid paraffin, provided that such paraffin shall be free from mineral acid.

Schultze Blasting-powder.—(As above defined.)

In the chemical report for the year 1892, Dr. Dupré reported adversely on the new variety of this powder which differed from the powder already

* Cotton gunpowder is now designated potentite (or tonite or cotton-powder No. 1).

licensed by the presence of one new constituent, which was to serve both as a gelatinizing and as a deadening agent. The new constituent was found to endanger the stability of the powder.

The following is an analysis of the powder under the 1884 definition given by Colonel Cundill :—

Soluble nitro-lignin	24·83
Insoluble nitro-lignin	23·36
Lignin (unconverted)	13·14
Nitrates of potassium and barium	32·35
Paraffin	3·65
Matters soluble in alcohol	0·11
Moisture	2·56

Schultze blasting-powder consists of Schultze gunpowder with the addition of charcoal.

Cooppal Powder.

This explosive was licensed in 1884, under Class 3, Division 2, and is a sporting powder, similar to Schultze powder. The following is the authorized definition which is extant at the present time :—

Cooppal Powder, consisting of nitro-lignin, carefully purified, with or without admixture of a nitrate or nitrates other than nitrate of lead and starch.

Espir Explosive Powder.

This powder was licensed in 1878, and described under Class 2 as consisting of a mechanical mixture of nitrate of sodium, sulphur, and sawdust. It does not seem to have been received with much favour by the public, in 1884 it was no longer manufactured, and disappeared from the authorized list in 1886.

The patent specification (No. 291, 1875) gives the following composition :—Nitrate of sodium, 60 parts ; sulphur, 14 parts ; and sawdust, 26 parts.

Asphaline.

This is one of the few chlorate mixtures, and was licensed under Class 4, in 1882 as follows, viz. :—

Asphaline No. 1, consisting of a mixture of chlorate of potassium and bran, as hereinafter defined, with or without an admixture of one or more of the following ingredients, viz. :—Nitrate of potassium, sulphate of potassium, paraffin oil, paraffin, ozokerit (such paraffin oil, paraffin, and ozokerit to be free from mineral acid), soap, fuchsine ; such mixture to contain not more than 54 parts by weight of chlorate of potassium and 4 parts by weight of nitrate of potassium and sulphate of potassium, or either of them, to every 42 parts by weight of bran. Bran consisting of wheat bran or barley bran, thoroughly cleansed, and reasonably free from flour.

Asphaline No. 2, consisting of asphaline No. 1, as above described, with the addition of nitrate of potassium, in such proportions that the total amount of nitrate of potassium present shall not exceed 25 parts by weight in 100 parts by weight of the finished explosive.

The following was the actual composition :—66 parts of chlorate of potassium and 34 parts of bran.

The manufacture of this explosive was discontinued in 1884, and in 1886, it was removed from the list.

Patent Safety Blasting-powder.

This is a nitrate mixture, licensed in 1881 under Class 2. Its definition has not been altered up to the present time, but the manufacture seems to have been suspended since 1884.

Patent Safety Blasting-powder, consisting of a mechanical mixture of saltpetre, sulphur, lampblack, sawdust, and sulphate of iron.

In the patent specification (No. 3934, 1874), it is defined as follows :—Nitrate of potash, nitrate of soda, nitrate of lime, one, two or three nitrates, in all 50 to 64 parts ; sulphur, 13 to 16 parts ; tanner's bark (containing refuse animal matter is preferred) or sawdust, or bark and sawdust, 14 to 16 parts ; soot or lampblack, or both, 9 to 18 parts ; and 5 to 6 parts of sulphate of iron in every 100 parts of the above mixture.

It is recommended to be used as a wash, with about 10 gallons of water to 2 pounds of the compound, as a remedy for *Phylloxera vastatrix*. (Colonel Cundill.)

Roburite.

Roburite is a nitro-compound and was authorized in 1887, under the following definition, Class 3, Division 2, viz. :—

Roburite, consisting of a mixture of nitrate of ammonium and thoroughly purified, chlorinated dinitro-benzole, provided that such chlorinated dinitro-benzole shall not contain more than 4 parts by weight of chlorine to every 100 parts by weight of the chlorinated dinitro-benzole.

During 1887, several modifications in the composition of roburite were sanctioned, and in 1888, the following was the authorized definition, which is still extant, viz. :—

Roburite, consisting of (*a*) nitrate of ammonium, with or without an admixture of nitrate of sodium and neutral sulphate of ammonium, or either of them, provided that the amount of nitrate of sodium shall in no case exceed 50 per cent. of the total amount of nitrates present ; and (*b*) thoroughly purified chlorinated dinitro-benzole, with or without the addition of thoroughly purified chloro-nitro-naphthalene and chloro-nitro-benzole, provided (1) that such chlorinated dinitro-benzole shall not contain more than 4 parts by weight of chlorine to every 100 parts by weight of chlorinated dinitro-benzole ; and (2) that the proportions of chloro-nitro-naphthalene and chloro-nitro-benzole shall not amount to more than 2 per cent. and 5 per cent. respectively of the finished explosive.

Roburite No. 2 appeared first on the authorized list of explosives in 1890, as follows :—

Roburite No. 2, consisting of roburite No. 1, as above defined, with the addition of chloride of ammonium and sulphate of magnesium, or either of them.

This explosive (1890) was said to contain between 5 and 12 per cent. of chlorinated dinitro-benzole containing not more than 4 per cent. of chlorine, mixed with nitrate of ammonium. Samples used appeared to contain 87 parts of nitrate of ammonium and 13 parts of chloro-dinitro-benzole, with traces of carbonate of ammonium and chloro-nitro-naphthalene.

Dr. P. P. Bedson analysed roburite as follows :—

Makers' mark	A. Dec. 13, 1889.	B. Sept. 24, 1890.
Moisture	1·14	0·66
Nitrate of ammonium...	83·35	86·30
Chloro-dinitro-benzene	11·67*	11·74†
Matter insoluble in ether or water ...	3·14	1·09
Fixed residue	0·70	0·21
	100·00	100·00

Carbo-dynamite.

In 1888, a license for a small quantity of carbo-dynamite was taken out, and as no further license was taken the explosive was not kept on the list. In 1889, it re-appeared under the following definition in Class 3, Division 1 :—

Carbo-dynamite, consisting of not more than 90 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 10 parts by weight of charcoal sufficiently absorbent in quality when mixed in the above proportions to prevent exudation of the nitroglycerine, whether with or without the addition of one or more of the following substances, viz. :—Nitrate of potassium, nitrate of barium, carbonate of sodium, and carbonate of ammonium, provided that the proportion of carbonate present shall not exceed $1\frac{1}{2}$ parts by weight in every 100 parts by weight of the finished explosive.

The above definition is still extant.

The patentee claims that by the addition of 3 to 5 parts of carbonized cork (in lieu of an equal amount of kieselguhr) to ordinary dynamite the latter also becomes capable of resisting the action of water. (Col. Cundill.)

Securit or Securite, Flameless Securite, Compressed Securite.

In 1888, securite was licensed for importation only, and appears as a nitro-compound in Class 3, Division 2, under the following definition:—

Securit, consisting of a mixture of nitrate of ammonium and thoroughly purified meta-dinitro-benzole.

In 1889, it was defined as :—

Securite, consisting of a mixture of nitrate of potassium or nitrate of ammonium with thoroughly purified meta-dinitro-benzole.

* Melted between 78 and 82 degs C. † Melted at 85 degs. C.

In 1889, flameless securite was added to the list.

Flameless Securite, consisting of securite as above defined with the addition of oxalate of ammonium.

This explosive disappeared in the following year (1890), being included in securite as below.

In 1890, securite only appears on the list as follows:—

Securite, consisting of a mixture of nitrate of potassium or nitrate of ammonium, with thoroughly purified meta-dinitro-benzole, with or without the addition of oxalate of ammonium.

In 1890, *stone* securite appeared to consist of 26 parts of dinitro-benzole and 74 parts of nitrate of potassium; and *coal* securite, a mixture of 80 parts of nitrate of ammonium, 17 parts of dinitro-benzole, and 3 parts of oxalate of ammonium.

In 1891, compressed securite appeared as follows:—

ompressed Securite, consisting of a mixture of nitrate of potassium and nitrate of barium (or either of them) with thoroughly purified nitro-cellulose and one or more of the following substances:—Thoroughly purified meta-dinitro-benzole, thoroughly purified dinitro-toluol, thoroughly purified nitro-naphthalene, thoroughly purified dinitro-naphthalene.

The above two definitions are extant at the present time.

Denaby Powder.

This explosive first appeared on the authorized list of 1891-2:—

Denaby Powder, consisting of compressed securite, as above defined, with the addition of charcoal.

Bellite.

This explosive was licensed for importation only in 1888, and it was entered under Class 3, Division 2, as follows:—

Bellite, consisting of a mixture of nitrate of ammonium and thoroughly purified meta-dinitro-benzole.

This definition is extant at the present time.

Bellite appears to be a mixture of about 80 parts of nitrate of ammonium and 20 parts of meta-dinitro-benzole.

Di-flamyr.

This explosive was licensed in 1888, and appears under Class 3, Division 2, up to the present time. It is defined as—

Di-flamyr, consisting of nitro-cellulose, thoroughly purified, mixed or impregnated with a nitrate or nitrates other than nitrate of lead.

Carbonite.

Carbonite was licensed in 1888, under Class 3, Division 1, the definition remaining unaltered up to 1892, when a slight alteration was made.

In 1888, it was composed as follows:—Nitroglycerine, 25 parts (with or without ½ part of sulphuretted-benzole); wood-meal, 40 parts; nitrate

of potassium and nitrate of sodium, either or both, 34 parts; and carbonate of sodium, ½ part.

In 1892, nitrate of barium and carbonate of lime were added as ingredients. The definition then appeared as follows :—

Carbonite, consisting of not more than 25 parts by weight of thoroughly purified nitroglycerine (with or without the addition of not more than ¼ part of sulphuretted benzole) uniformly mixed with 75 parts by weight of a pulverized preparation consisting of wood-meal not less than 40 parts; nitrate of potassium, nitrate of sodium, and nitrate of barium (one or more of them) 34 parts; and carbonate of sodium and carbonate of lime (or either of them) not more than ½ part, such preparation to be sufficiently absorbent when mixed in the above proportions to prevent exudation of nitroglycerine.

The makers state the composition of carbonite to be as follows :—Nitroglycerine, 25 parts; nitrate of potassium, 30 parts; nitrate of barium, 4 parts; wood-meal, 40 parts; and carbonate of sodium, 1 part.

Favier Explosive, Miner's Safety Explosive, or Ammonite.

Favier explosive was licensed in 1888 under the designation of miner's safety explosive, and its name changed to ammonite in 1889.

In 1890, the composition was :—10 parts of mononitro-naphthalene and 90 parts of nitrate of ammonium.

In 1892, dinitro-naphthalene was added, and the definition at present stands under Class 3, Division 2, as follows :—

Ammonite, consisting of a mixture of nitrate of ammonium, and thoroughly purified mononitro-naphthalene or dinitro-naphthalene, or a mixture of the two last-named ingredients, made up into cartridges for blasting not containing their own means of ignition, contained or enclosed in a thoroughly waterproof case.

In 1893, a sample was analysed as follows :—

	Per Cent.
Dinitro-naphthalene	12·54
Nitrate of ammonium	87·46
	100·00

Fortis Explosive.

Fortis explosive was licensed in 1888 under Class 2, and defined as follows, viz. :—

Fortis Explosive, consisting of a mixture of two or more of the following substances, viz. :—Tan, lampblack, and sulphur, such mixture being thoroughly impregnated with a mixture of nitrate of potassium and proto-sulphate of iron, and with or without the addition to such impregnated mixture of glycerine,

This explosive is authorized for importation and issued only in the form of compressed cartridges, such cartridges being rendered thoroughly waterproof (*a*) by waterproofing the naked compressed cartridges and (*b*) by enclosing such waterproof compressed cartridges in a thoroughly waterproof cartridge-case. The above definition is extant at the present time.

The component parts are :—Sulphur 12, charcoal 11, ferrous sulphate 5, nitrate of potassium 62·5, moisture 2·5, and organic sulphide 7.

Fortisine.

This explosive is also called Fortis No. 2, and was licensed in 1892, under Class 3, Division 2, as follows, viz.:—

Fortisine, consisting of a mixture of saltpetre, sulphur, and charcoal with the addition of a dinitro-benzole and resin or dextrine, provided, 1st, that the amount of dinitro-benzole shall not exceed 5 per cent. of the finished explosive; and 2nd, that all the ingredients shall be thoroughly purified.

Ballistite.

Ballistite is a blasting-gelatine, licensed in 1889; and a slight alteration was made in the definition in 1890. It appears in 1892 under Class 3, Division 1, and is defined as follows :—

Ballistite, consisting of nitro-cellulose, carefully washed and purified, combined with thoroughly purified nitroglycerine, with or without the addition of camphor, aniline, and such other substance, and solvent (if any) as may from time to time be approved by a Secretary of State, and with or without carbonate of calcium or carbonate of magnesium not exceeding 1 part by weight in every 100 parts by weight of the finished explosive, the whole to be of such character and consistency as not to be liable to liquefaction or exudation.

It is composed of from 2 to 1 parts of nitroglycerine, with from 1 to 2 parts of nitro-cellulose.

Stonite.

Stonite was licensed in 1889. The definition, under Class 3, Division 1, has remained unaltered since then, as follows :—

Stonite, consisting of not more than 68 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 32 parts by weight of a preparation consisting of nitrate of barium, nitrate of potassium (or either of them), kieselguhr (not less than 20 parts by weight), wood-meal (not less than 4 parts by weight), and carbonate of magnesia, with or without the addition of sulphuretted oil and soot (or either of them), such preparation to be sufficiently absorbent when mixed in the above proportions to prevent exudation of nitroglycerine.

It is composed of 68 parts of nitroglycerine, and charcoal, kieselguhr, and wood-meal 32 parts.

Picric Acid.

Picric acid was licensed as an explosive in 1889 (with no alteration up to the present time), and defined under Class 3, Division 2, as follows :—

Picric Acid, consisting of trinitro-phenol containing not more than 0·5 per cent. of mineral matter or ash.

Picric acid or trinitro-phenol ($C_6H_3(NO_2)_3O$) is obtained by the action of nitric acid on carbolic acid. It is a yellow crystal used largely as a dye. When heated in contact with certain metallic salts or oxides, it explodes violently. It is seldom if ever used as a blasting agent owing to the

large percentage of carbonic oxide given off. Owing to a disastrous explosion at Messrs. Roberts, Dale & Co.'s chemical works, near Manchester, in 1887, picric acid was brought within the scope of the Explosives Act, with certain exceptions.

Gathurst Powder.

Gathurst powder appeared on the 1890 list, without comment, as a new explosive, under Class 3, Division 2, and is extant at the present time.

Gathurst Powder, consisting of a mixture of (*a*) nitrate of potassium or nitrate of sodium, with or without the admixture of neutral sulphate of ammonium, nitrate of ammonium, chloride of ammonium, sulphate of magnesium, and a colouring matter, such as charcoal or lampblack (free from mineral acid), or all or any of them; and (*b*) thoroughly purified nitro and chloro-nitro compounds of benzine, toluene, and naphthalene, with or without such other substances as may from time to time be approved by a Secretary of State. Provided that the finished explosive shall not contain more than 2 parts by weight of chlorine in every 100 parts of the finished explosive.

The composition is chloro-dinitro-benzole 1 part, nitrate of potassium 4 parts, nitrate of ammonium 2 parts, and colouring matter *ad lib.*

Smokeless Powder and Smokeless Blasting-powder.

Smokeless powder was licensed in 1888, and smokeless blasting-powder in 1891, under Class 3, Division 2, and they are defined at the present time (1892) as follows:—

Smokeless Powder, consisting of nitro-lignin, carefully purified and mixed or impregnated with a nitrate or nitrates (other than nitrate of lead and ammonium nitrate), and with or without starch or collodion, or turmeric or similar vegetable colouring matter, provided that such collodion shall consist of carefully purified nitro-lignin dissolved in a safe and suitable solvent, and with or without such other substance as may from time to time be approved by a Secretary of State.

Smokeless Blasting-powder, consisting of smokeless powder, as above defined, with the addition of any one or more of the following ingredients, viz.:—Dinitro-benzole, dinitro-toluol, nitro-benzole, and nitro-naphthalene, provided that all such ingredients shall be thoroughly purified.

Its composition is nitro-cotton 65 parts, nitro-toluol 9 parts, nitrate of barium 20 parts, nitrate of potassium 4 parts, and chalk 2 parts.

Forcite.

This explosive was licensed in 1891, under Class 3, Division 1, and was defined as under:—

Forcite, consisting of thoroughly purified nitroglycerine, thickened by being combined with nitro-cellulose, carefully washed and purified, and mixed or incorporated with wood-meal and nitrate of potassium, in such proportions that the whole shall be of such character and consistency as not to be liable to liquefaction or exudation.

The above definition is extant at the present time.

Oarite.

Oarite was licensed in 1891, under Class 3, Division 1, and was defined as follows :—

Oarite, consisting of not more than 2 parts by weight of thoroughly purified nitroglycerine, uniformly mixed with 8 parts by weight of the following preparation, viz. :—Thoroughly purified nitro-cellulose 1 part, thoroughly purified dinitro-benzole 1 part, nitrate of barium or nitrate of potassium, or either of them, 6 parts, provided that the said preparation shall be sufficiently absorbent in quality when mixed in the above proportions to prevent exudation of nitroglycerine.

The above definition is extant at the present time.

Ardeer Powder.

Ardeer powder was licensed in 1891 under Class 3, Division 1, and was defined as follows, viz. :—

Ardeer Powder, consisting of dynamite No. 1, as above defined,* mixed or incorporated with sulphate of magnesium.

In 1892, an alkaline nitrate was added to the powder, and it appears on the list at the present time as follows, viz. :—

Ardeer Powder, consisting of dynamite No. 1, as herein defined,* mixed or incorporated with sulphate of magnesium, with or without the addition of nitrate of potassium, nitrate of sodium, nitrate of barium, or such other nitrates as may from time to time be approved by the Secretary of State.

The composition of ardeer powder is defined by the makers as follows : —Nitroglycerine $33\frac{3}{4}$ parts, charred kieselguhr $11\frac{1}{4}$ parts, and sulphate of magnesium and nitrate of potassium 55 parts.

Blasting Matagnite and Matagnite-gelatine.

These explosives appeared on the 1891 list, under Class 3, Division 1, and are defined at the present time as follows :—

Blasting Matagnite, consisting of nitro-cellulose, carefully washed and purified, combined with thoroughly purified nitroglycerine and thoroughly purified nitro-benzole, or either of them, in such proportions that the whole shall be of such character and consistency as not to be liable to liquefaction or exudation.

Matagnite Gelatine, consisting of thoroughly purified nitroglycerine and thoroughly purified nitro-benzole, or either of them, thickened by being combined with nitro-cellulose, carefully washed and purified, and mixed or incorporated with wood-meal, for the bleaching or purifying of which no chemical agents have been used, or, if used, have been absolutely removed, and nitrate of potassium or such other nitrate as may for the time being be sanctioned by a Secretary of State, in such proportions that the whole shall be of such character and consistency as not to be liable to liquefaction or exudation.

It is said to consist of nitroglycerine 65 parts, nitro-cellulose 4 parts, nitrate of potassium 24 parts, and wood-meal 7 parts.

* See page 157.

Greener Powder.

This explosive was licensed in 1891, under Class 3, Division 2, as follows :—

Greener Powder, consisting of a mixture of thoroughly purified nitro-cellulose with thoroughly purified nitro-benzole, or either of them, with the addition of colouring matter consisting of graphite, lampblack, or other suitable material, such colouring matter to be free from free mineral acid.

Cannonite Nos. 1 and 2.

These explosives were licensed in 1892, under Class 3, Division 2, and were defined as follows :—

Cannonite No. 1, consisting of gun-cotton (consisting of thoroughly purified nitro-cellulose) mixed or impregnated with a nitrate or nitrates (other than nitrate of lead or nitrate of ammonium) and resin, and with or without the addition of graphite.

Cannonite No. 2, consisting of gun-cotton (consisting of thoroughly purified nitro-cellulose) mixed or impregnated with resin, and with or without the addition of graphite.

Troisdorf Smokeless Powder.

This explosive was licensed in 1892, under Class 3, Division 2, and was defined as follows :—

Troisdorf Smokeless Powder, consisting of thoroughly purified nitro-cellulose gelatinized by a suitable process, and with or without the addition of nitrates (other than nitrate of ammonium).

Amberite Nos. 1 and 2.

Amberite No. 1 was licensed in 1892, under Class 3, Division 1, as follows :—

Amberite No. 1, consisting of thoroughly purified nitro-cellulose mixed or combined with the following substances :—Thoroughly purified nitroglycerine, paraffin free from mineral acid, and shellac. The whole to be of such character and consistency as not to be liable to liquefaction or exudation.

Amberite No. 2 was licensed in 1892, under Class 3, Division 2, as follows :—

Amberite No. 2, consisting of thoroughly purified nitro-cellulose, whether or not mixed or impregnated with nitrate of potassium and nitrate of barium, or either of them, and with or without the addition of paraffin, vaseline, and graphite, or such other substance as may from time to time be approved by a Secretary of State

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The meeting then closed.

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,

DECEMBER 9TH, 1893.

MR. A. L. STEAVENSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings on November 25th and that day.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. FRANK J. AGABEG, General Manager, Apcar & Co.'s Collieries, Sitarampore, India.

Mr. JOHN ALEXANDER CHALMERS. Mining Engineer, P.O. Box 357, Johannesburg, Transvaal.

Mr. JOSEPH CHATER, Mining Engineer, Bengal Nagpur Coal Co., Limited, Evelyn Lodge, Asansol, Bengal, India.

Mr. FRANCISCO M. COGHLAN, Engineer, Catorce S.L.P., Mexico.

Mr. F. LAWRENCE CORK, General Manager, Bengal Nagpur Coal Co., Limited, Barakur, Bengal, India.

Mr. THOMAS CROUDACE, Colliery Manager, West House, Haltwhistle.

Mr. JOHN ETHERINGTON, Consulting and Mechanical Engineer, 39A, King William Street, London, E.C.

Mr. CHARLES FERGIE, Mining Engineer, Drummond Colliery, Westville, Nova Scotia.

Mr. JAMES FLETCHER, Colliery Manager, Wickham and Bullock Island Collieries, Carrington, New South Wales.

Mr. EDWARD GLEDHILL, Mining Engineer, Carolina Hacienda, Honda, Republic of Colombia, South America.

Mr. William Griffith, Manager, The De Beers Prospecting Syndicates, Waterloo House, Aberystwyth, South Wales, and Fort Salisbury, Mashonaland, South Africa.

Mr. John Ernest Hardman, Mining and Consulting Engineer, P.O. Box 520, Halifax, Nova Scotia.

Mr. Robert George Higby, Mining Engineer, Manager, Borrea Coal Co., Limited, Sitarampore, Bengal, India.

Mr. Robert G. Leckie, Mining and Metallurgical Engineer, General Manager, Londonderry Iron Co., Limited, Londonderry, Nova Scotia.

Mr. Alfred Benjamin Lindop, Mining Engineer, Blackball, *via* Greymouth, New Zealand.

Mr. George Hamilton Lloyd, Civil and Mining Engineer, Dolgerddon, Rhayader, Radnorshire.

Mr. Carr Waller Pritchett, Jun., Mining Engineer, Apartado 84, Pachuca, Estado Hidalgo, Mexico.

Mr. C. Robinson, Mining Engineer, c/o Messrs. Wilmard, Spillhouse, & Co., Cape Town, South Africa.

Mr. Alfred F. Seecombe, Mining Engineer, Albaston, Gunnislake, Cornwall.

Mr. Charles William Sibold, Civil Engineer, Public Works Department, Bengal Civil Service, Mirzapore, N.W.P., India.

Mr. Peregrine Oliver Wilson, Mining Engineer, 30, Finsbury Circus, London, E.C.

Associate Member—

Mr. Henry Davies, Lecturer in Geology and Mining for the County of Glamorgan, Treharris, R.S.O., South Wales.

Associate—

Mr. Ralph B. Dormand, Under Manager, Waldridge Colliery, Chester-le-Street.

Students—

Mr. Wardle Asquith Swallow, Mining Student, Bushblades House, Lintz Green.

Mr. Thomas James Tomlinson, Mining Student, Harton Colliery, South Shields.

The following gentlemen were nominated for election :—

Members—

Mr. Thomas Adamson, Assistant Manager, E.I.R. Collieries, Bengal, India.

Mr. Walter Bell, Colliery Manager, 23 Windsor Terrace, Newcastle-upon-Tyne.

Mr. Charles Butters, Engineer, Rand Central Ore-reduction Co., P.O. Box 1891, Johannesburg, Transvaal.

Mr. Thomas Carr, Colliery Manager, Bengal Iron and Steel Co., Limited, Colliery Manager's Office, Burrakur, East Indian Railway, Bengal, India.

Mr. Richard E. Chism, Mining Engineer, 3A, Independencia No. 1, City of Mexico.

Mr. Napier Cochrane, Colliery Manager, Aldin Grange, Durham.

Mr. G. D. Delprat, Mining Engineer, c/o Messrs. Hill, Delprat, Ferdinand, & Carr, Campañas No. 15, Cordoba, Spain.

Mr. Fred W. Hall, Colliery Manager, Haswell Lodge, Sunderland.

Mr. George E. Harris, Mining Engineer, Margherita, Debrugarh, Upper Assam.

Mr. J. Howard Johnston, Metallurgical Engineer, Establecimiento Minéral de Casapalca, Lima, Peru.

Mr. Joseph Cook Nicholson, Manufacturer of Engineers' Tools, 59, Side, Newcastle-upon-Tyne.

Mr. R. Price-Williams, Civil Engineer, Victoria Mansions, 32, Victoria Street, London, S.W.

Mr. James Prout, Mechanical Engineer, 16, Claremont Road, Redruth, Cornwall.

Mr. Francis Bell Forsyth Rhodes, Superintendent, National Smelting and Refining Co., South Chicago, Illinois, U.S.A.

Mr. Charles Robinson, Mine Owner and Prospector, Steries Cottage, Marias Road, Cape Town, South Africa.

Mr. Alexander Smart, Assistant Engineer, De Beers Mine, Kimberley, South Africa.

Mr. Frank R. Sykes, Colliery Manager, Springwell Villa, Bishop Auckland.

Mr. Joseph Woodburne, Mining Engineer, Kimberley Mine, Kimberley, South Africa.

Associate Members—

Mr. Archibald Blue, Director, Bureau of Mines, Toronto, Canada.

Mr. John Goethe East, Ship Share Broker, 26, Side, Newcastle-upon-Tyne.

Mr. James I'Anson, Fairfield House, Darlington.

Mr. Arthur C. Payne, Royal College of Science, South Kensington, London, S.W.

Associates—

Mr. Michael Heslop, Under Manager, Rough Lea Colliery, Willington.

Mr. Thomas Wallett, Deputy Overman, Heworth Colliery, Felling, R.S.O., Durham.

Student—

Mr. Frederic Lancelot Booth, Mining Student, Ashington Colliery, Morpeth.

DISCUSSION ON MR. D. MURGUE'S PAPER ON "THE FRICTION OF, OR RESISTANCE TO, AIR-CURRENTS IN MINES."*

The PRESIDENT said that Mr. Murgue's paper was very instructive, and recorded an example of carrying out very delicate experiments in a careful manner. For over thirty years he had accepted the statement of the late Inspector of Mines for the South Durham district—Mr. J. J. Atkinson, who had gone very carefully into the matter. That gentleman took the value of the coefficient of friction in the galleries of a coal-mine at 0·00417, and this had always been accepted as being at all events sufficiently accurate for practical purposes. Mr. Murgue, however, whom they all knew to be a very able authority on questions of mine ventilation, had made a series of experiments upon the length of airways not greatly exceeding (except in one instance) 300 feet. He had carried out the experiments by using a delicate form of water-gauge, whose readings could be observed, by means of a microscope, to 0·0008 inch, and he had by these means ascertained that the coefficient of friction for various kinds of passages varied from 0·00030 to 0·00241. This last value was just about one-half of the value recommended for use by Mr. Atkinson, but he (the President) was strongly under the impression that if they took all the galleries of a colliery between the downcast and the upcast shaft they would find that, although Mr. Atkinson erred on the safe side, the error was not of any great extent. And although Mr. Murgue gave excellent data, he (the President) would not recommend any one who had to erect a fan or who had to go seriously into the question of the friction of the mine to take any value very much less than that recommended by Mr. Atkinson. In conclusion, he would recommend the members to study Mr. Murgue's valuable paper, which he thought was of a most admirable and useful character.

Mr. T. L. ELWEN said that he had been very much interested in perusing Mr. Murgue's paper, with its record of elaborately conducted experiments. He had no wish to dispute the accuracy of the values of the coefficient of friction recorded in the paper as a guide in calculating the amount of pressure lost on current resistance in the varying types of airways. As, however, he (Mr. Elwen) had made this question the subject of a large number of experiments, some of which are recorded in the *Transactions*,† he might be excused for offering a few remarks, such as occurred to him while reading the paper. The statement, "that variations

* *Trans. Fed. Inst.*, vol. vi., page 135.

† *N. of Eng. Inst.*, vol. xxxviii., page 205.

of the volume of air did not at all modify the configuration of the curves of equal velocity"* was a surprise to him, and much more so was the difference between the maximum and minimum velocities of the air recorded in the tables. There was a very great difference between the maximum and minimum velocities, varying from 32 per cent. in experiment No. 10 to 435 per cent. in experiment No. 4. Mr. Murgue did not state what is the least distance from the sides at which the anemometers were held, and his impression was that timber or a jagged piece of rock was in the immediate vicinity, at any rate, in those examples which showed such a great difference. The result of special experiments which he had made by dividing into squares by strings two airways, (one square and the other circular), the nearest distance from the sides at which the velocity was taken being 1 foot, showed that for a low current-velocity in the square airway, the divisions in the lower half of the airway gave the highest velocities, in fact, in one instance the vanes of the anemometer would turn only near the bottom, the lowest reading not exceeding 40 per cent. below that at the centre. For a high current-velocity the readings at the sides more generally approached that of the centre. For a circular airway these percentage amounts were much smaller. In his experiments he had been unable to find that the rule adopted by Mr. Murgue was true, and would prefer to adhere to the system of squares for correct measurement of the volume of air. He was doubtful whether he could accept the coefficients of friction based on Mr. Murgue's system of deducing the current-velocity. Mr. Murgue did not state whether a fan or a furnace was the ventilating power employed, and in his (Mr. Elwen's) opinion the latter is very irregular and renders the deductions from comparative velocities valueless. A record of the revolutions of the fan—if such was used—during the period of such air-measurements was very desirable. He did not agree with Mr. Murgue's deductions on the same grounds as were expressed at the former discussion on this paper, namely, that the airways experimented on were too short to embrace a sufficient average working length of uniform area. The inevitable presence of refuge-holes, stenton-ends, etc., even in a regular sectional area, would modify the amount of resistance given by the author as for airways of regular section. He had already pointed out the effect of these interruptions to the regular course of an air-current in a paper read before the members, and he ventured to assert that had Mr. Murgue taken longer lengths of airway he would have found an increased amount of resistance over and above that which proportionately would have resulted from an

* *Trans. Fed. Inst.*, vol. vi., page 143.

increase in length. Mr. Deacon had referred* to a very high value of the coefficient of friction found in an experiment (made with the pit top-doors open) in which the air had a free course for a short length of the upcast shaft. He thought that this value was incorrect, and that it included the amount of power necessary to create the velocity, as well as the direct current resistance.

Mr. M. Walton Brown said the distance at which the anemometers were placed from the side of the airway in Mr. Murgue's experiments was shown in each of the diagrams, which were all drawn to scale. All of the experiments were made at the Bessèges and Créal Collieries, both of which were ventilated by means of centrifugal fans. The great difficulty in making any experiment upon the friction of air in mine galleries was that of obtaining a sufficient length of gallery possessing a sufficient regularity of area and of the nature of the sides.

The President said he did not think it should be difficult in this district to get a suitable place for experiments, and perhaps some of the members might be able to offer the use of suitable galleries for further experiments by Mr. Elwen.

The discussion was then closed.

Mr. J. P. Kirkup read the following paper on the "Singareni Coal-field, Hyderabad, India" :—

* *Trans. Fed. Inst.*, vol vi., page 175.

SINGARENI COAL-FIELD, HYDERABAD, INDIA.

By J. P. KIRKUP.

Discovery, Situation, and History.

The Singareni coal-field was discovered by Dr. King in the year 1872 while examining the geology of that part of the Deccan.

Subsequently, this field was partly proved by borings put down by officers of the Nizam's Government, and the results obtained were so satisfactory that a small trial-shaft was sunk, and a few hundred tons of coal taken from near the outcrop for trial in locomotive engines at Hyderabad. Nothing further was done until 1886, when steps were taken to continue the line of railway from Hyderabad to Bezwada, and to tap the field by a branch line.

The mining rights in the field and a monopoly to prospect the whole of the Nizam's dominions, an area of 81,000 square miles, for a period of four years, were granted to a syndicate, who formed the Hyderabad Deccan Company, with a nominal capital of £1,000,000, of which only £150,000 was subscribed, the balance of unsubscribed shares being held by the concessionaires.

Subsequently a Parliamentary enquiry was instituted to protect the native Government, and the concessionaires agreed to subscribe another £150,000 in deferred shares to ensure the confirmation of the concession. A considerable amount of capital has been expended on prospecting work at the so-called Golconda diamond mines (situated on the banks of the Kristna river), which have proved unsuccessful, and also in small areas of gold-bearing schists in the south-western part of the territory. Some old workings have been partially explored, but up to the present time no very rich quartz has been found.

The remainder of the territory has been practically unprospected by the company.

Active work commenced in 1887 under Mr. Hughes, superintendent of the Indian Geological Survey, with a staff of European miners. In the early stages much difficulty was experienced owing to the ground being covered with thick jungle ; only one outcrop was visible, and the whole of the labour was imported. Owing to want of proper shelter, bad food supplies, and bad water, cholera and other diseases swept the place, causing at one time nearly the whole of the people to decamp.

Mr. Hughes was subsequently able to much improve matters by instituting order and cleanliness and improving the water-supply.

The coal-field is situated in about North latitude 17 degs. 30 mins. to 17 degs. 40 mins., and East longitude 80 degs. 18 mins. to 80 degs. 25 mins., and is connected with the standard gauge Indian lines by the Nizam's State Railway.* It is 653 miles from Bombay, the great port of the western side of India, and 146 miles from Masulipatam on the eastern coast. The latter can hardly be considered a port, as ocean steamers are unable to shelter and have to lie out some distance from the shore. Madras when connected by the east coast railway, at present under construction, will be about 360 miles distant, but by the present route it is 695 miles. The Indian Government have not yet decided to construct this part of the line, but are pushing on with the northern section towards Calcutta.

Bezwada is a rising town on the north bank of the river Kristna, in British territory. It is 96 miles distant from the coal-field, and forms the junction of the east coast railway with the Nizam's State Railway and the narrow-gauge system of the Southern Maratta Railway. A fine bridge has been constructed by the Government of India across the Kristna river, to effect the junction of these railways and continue the east coast line northward towards Calcutta. The Godaverry river will be bridged shortly.

Bezwada is also situated on the system of canals, constructed under the direction of the late Duke of Buckingham for irrigation and light traffic. They connect the Godaverry river with the small port of Coconda, and also with Bezwada and Masulipatam and, crossing the Kristna river at the former town, are continued to Madras. These canals irrigate an enormous area of country and serve the purpose of a very large goods traffic during the wet season, from June to February. During the intervening dry months there is no water in them, and opportunity is then taken to remove the annual accumulations of silt.

Hyderabad, the capital of H.H. the Nizam's dominions, and Secunderabad, the British military cantonment adjacent to it, are the only towns of any consequence upon the line of railway in the Nizam's dominions. The only coal-consuming industries situated there, are a cotton-mill and an ice-factory, with brick and lime-burning carried on to a small extent.

* The position of the Singareni Coal-field and its railway connexions is shown on plate IV. *bis*, vol. xxxvii., *Trans. N.E. Inst.*

Gulbarga is another town where a cotton-mill has been established. Several smaller towns on the line of railway are shaking off the sleep of ages and becoming busy centres of trade in grain, oil-seeds, wool, and cotton, but no manufacturing industries, save the three previously mentioned, have been attempted.

Lack of security and want of confidence by the natives in the native Government have much to do with this, together with an objection to European enterprise within the State.

Physical Features.

The district of Telingana, in which the coal-field is situated, was once considered the "garden of India," but the disorder following its conquest by the Mahomedans converted it into a wilderness, from which, however, it is now gradually emerging under the more enlightened rule of H.H. the present Nizam.

The country is overgrown by dense jungle, and the traveller sees with regret merely the evidences of former cultivation. Huge tanks, formed by the construction of dams across small streams, have been allowed to fall into ruin, and the fertile lands, once well watered, are hardly visible, owing to the jungle which covers them.

A scanty population of aborigines procure their subsistence from small clearings, upon which they grow various kinds of pulse, millet, and maize.

The surface of the coal-field is gently undulating and encircled by craggy hills of primitive rock, the surrounding country being a most picturesque combination of hill and valley. The general elevation of the coal-field is about 700 feet above sea-level, and the surrounding hills are from 500 to 600 feet higher. The drainage is effected by three rivers, one from the northern extremity, one in the centre, and the third drains the southern portion of the field. The district is generally dry, even in the rainy season. The climate, though hot and relaxing, is fairly healthy during the hot months and throughout the rainy season, but during the months of November, December, and January malarial fever becomes very prevalent among Europeans and natives alike. This is no doubt due to the immense jungles of the district.

The temperature during the months of March, April, and May often rises to 106 degs. Fahr. in the shade, and seldom falls below 90 degs.

The rainfall is about 30 inches per annum ; it usually commences in the second week of June and finishes in September.

The quality of the water is bad, being generally brackish, and during

the rainy season it causes a severe form of scorbutic ulcer upon the legs and feet of the people. A small scratch will open out and spread into an enormous ulcer.

The inhabitants are chiefly aboriginal tribes called Koyavas. They are a quiet, inoffensive people, noted for honesty, are splendid woodmen, and with light axes of their own manufacture fell the hardest timber with consummate ease.

At the commencement of mining operations they were extremely timid and could not be induced to come to even the surface-workings, but afterwards they came forward in considerable numbers, and many who have taken to mining are efficient coal-cutters.

Another tribe, of wandering propensities, inhabits the jungles ; they are the Benjaries or Indian gipsies. They are possessed of large herds of cattle, and were originally, before the days of railway traffic, the great carriers of the country, grain being even now carried hundreds of miles upon the backs of their cattle. They are an athletic, fine race, and noted thieves, but the march of modern civilization is compelling them to find settled employment. The more open parts of the district are peopled by various castes of Hindoos, the official classes being Mahomedans. The villages are chiefly peopled by cultivators and small traders. The cultivator class furnishes the greater part of the labour at the collieries.

Geology.

The Singareni coal-field (Fig. 1, Plate XII.) is a small outlier of the great deposit of Indian Coal-measures found in the Godaverry valley. It includes representative beds of all the formations of the Lower Gondwana system, and occupies an area of about 26 square miles.

Indian geologists have divided the formations there found into Upper and Lower Gondwana,* and these are further subdivided as follows :—

Upper Gondwana—
- Chikiala sandstones.
- Kata sandstones.
- Maliri sandstones.

Lower Gondwana—
- Kamthis sandstones.
- Barakur beds.
- Talchir beds.

Vindyan—
- Metamorphic schists.
- Crystalline limestones.

Gneiss.

Fig. 2, Plate XII., shows the structure of the coal-field and the relation of its three geological divisions.

* These measures are the equivalent of the Triassic formation.

Coal is found in the Barakur beds of the Lower Gondwana group, and in the Singareni field the three formations of that group are all very distinctly shown: in this respect it resembles the larger neighbouring area of the Godaverry valley.

It appears at parts of the boundary as though the Singareni field had been faulted into a trough of the earlier rocks, and thus preserved from influences which would otherwise have denuded it previous to the deposition of the Kamthis beds. Evidences of faulting are plainly visible on both the north-eastern and south-western boundaries of the coal-field, where the Gondwana group abuts against the crystalline Vindyan and Gneiss-rocks, although at the centre of the eastern boundary a natural contact is evident. The Barakur rocks are there found lying naturally upon the Gneiss, as proved by borings.

The Kamthis rocks have evidently suffered great denudation at a later period; with the exception of a solitary hill about 500 feet high on the north-eastern edge of the field, they are not of greater thickness than 100 feet, and in places nearly the whole has been denuded and the Barakur rocks exposed on the surface.

The Kamthis beds of the hill referred to overlap the Barakur formation and rest upon Gneiss.

The upper beds of the Kamthis group, as exhibited in the hill, very much resemble the Barakur rocks, being light grey sandstones, with light shaly beds intermixed; they are, however, of a more coarse and gritty character, the quartz-grains being bound together by decomposed felspar. The lower beds of the Kamthis group have, however, a more distinct character, and consist of very coarse yellow, brown, and red sandstones, with thin beds of very pure white clay or kaolin. These rocks exhibit great irregularity of bedding, having a very irregular dip, and often showing the planes of separate beds running into each other. The thickness and character of these beds vary infinitely, even in very short distances.

Boreholes within 200 feet of each other on the line of dip never showed the same section, as will be seen from the sections on the following page. B is 400 feet to the dip of A, and D is 200 feet to the dip of C, the surface being about level.

The vivid coloration of the rocks is evidently due to the presence of iron, as some of the lower beds are highly ferruginous, and were formerly made use of by the natives for the manufacture of iron. A pisolitic ore formed on the surface by the joint action of rain and sun was gathered for the purpose.

A.

	Ft.	Ins.
Red soil	6	0
Kamthis—		
Yellow sandstone ...	17	0
Red sandstone	9	0
Yellow sandstone ...	24	0
Yellow sandstone ...	9	0
Yellow sandstone ...	7	6
Dark variegated clay ...	2	6
Yellow sandstone ...	39	0
Barakur—		
Grey sandstone... ...	—	

B.

	Ft.	Ins.
Red soil	4	0
Kamthis—		
Decomposed sandstone	6	0
Yellow sandstone ...	12	0
Variegated clay... ...	2	0
Red sandstone	3	6
Yellow sandstone ...	57	6
Variegated clay... ...	6	0
Barakur—		
Grey sandstone	—	

C.

	Ft.	Ins.
Dark soil	3	0
Kamthis—		
Morum	4	0
Yellow sandstone ...	28	6
Variegated clay	2	6
Red sandstone	2	0
Variegated clay... ...	4	0
Yellow sandstone ...	7	0
Yellow clay	2	6
Barakur—		
Grey sandstone... ...	—	

D.

	Ft.	Ins.
Dark soil	2	0
Kamthis—		
Morum	7	0
Variegated sandstone ...	5	0
Yellow sandstone ...	48	0
Variegated clay... ...	2	0
Variegated sandstone ...	2	0
Variegated clay... ...	4	0
Variegated sandstone ...	4	0
Barakur—		
Grey sandstone... ...	—	

The outcrops of the Kamthis rocks usually weather into round boulder-looking forms with a tendency to disintegrate from the parent bed.

At Singareni, no fossil remains have been found in the Kamthis beds, elsewhere *Calamites* occur, but they are rare, and although these beds are classed by the Geological Survey as Kamthis, they may be Barakur rocks.

The Barakur, or coal-bearing formation of the field, is almost entirely covered by the unconformable Kamthis rocks (Figs. 2 and 3, Plate XII.). A small area on the eastern side, about the centre of the field, shows that the Barakur formation crops out in the river, adjoining a boundary of gneiss; and a narrow outcrop of the same beds is seen on the western boundary immediately opposite, abutting against Vindyan rocks.

The only outcrop of coal upon the field was that found by Dr. King in a pothole in the rocky bed of the river at the point *c*, indicated upon Fig. 1, Plate XII., on the eastern side of the field.

The general dip of the measures is south-westerly, being from 6 to 8 inches per yard at the outcrop, but it is probable that it will become flatter towards the centre of the basin.

The measures consist of light grey sandstone and dark shaly beds alternating with beds of coal. The sandstones are finer-grained than and quite distinct from those of the Kamthis series found above, the Barakur sandstones bearing a strong resemblance to some of those of the English Coal-measures. The bedding and general character of the Barakur rocks are much more regular than the Kamthis group, and the outcrops are easily distinguished by their more regular bedding, sharp edges, and a tendency to wash into potholes under the scour of a river-current. No distinguishing fossils have been found, but a few impressions of *Calamites* are occasionally seen on the roof of the King seam.

Another circumstance noted is that the soil overlying the Barakur beds is frequently impregnated with lime, which separates out in the form of nodules of Kunker limestone.

The general thickness of the Barakur beds is from 500 feet to 600 feet, as proved by borings to various depths at different horizons on the outcrop. The outcrop-edges of the seams abutting against the overlying Kamthis rocks are of a muddy, soft, peaty nature, which rapidly passes into good coal under the cover of Barakur rocks. These appearances indicate that the outcrop-edges of the seams have been subjected to considerable weathering agencies previous to the deposition of the Kamthis group.

The following may be taken as a characteristic general section of the coal-field :—

Name of Formation.	Description of Strata.	Thickness.
		Feet.
Kamthis ...	Yellow, brown, and red sandstones and clays	80
Barakur ...	Grey sandstone and shales	160
	Thick coal-seam	30 to 40
	Grey sandstones, shales, and thin coaly beds	100
	Coal-seam	2 to 5
	Grey sandstone...	35
	Coal-seam	2 to 5
	Grey sandstone...	40
	King coal-seam...	3 to 7
	Grey sandstone...	100
	Coal-seam	1 to 2
	Grey sandstone...	110
Talchir ...	Muddy, dun, and buff sandstone	200

Notwithstanding the fact of the Barakur rocks being generally hidden by the overlying Kamthis beds, various borings have proved them to extend over an area of about 12 square miles, and of that area about 9 square miles are coal-bearing.

The Talchir beds which are well developed, and seen on the surface at the north-eastern end of the coal-field, consist of well-marked sandstones and shaly mudstones of a dirty brown or dun colour. They lie at low angle upon the Vindyan crystalline rocks, and dip south-west underneath the Kamthis and Barakur groups to which they are unconformable. No coal has been found in these rocks in Southern or Central India; indeed they may be termed the farewell rock of the Indian Coal-measures, and are known as such if met with in boring for coal. At their base is a bed of conglomerate formed of fragments of the earlier rocks. This conglomerate shows evidence of glacial action, as observed by Mr. Fedden on the banks of the Pem river, south-west of Chanda, in the Central Provinces. It has a very small development at Singareni, but is of importance as showing the similarity of these Talchir rocks to those of other parts of India. Large sharp-edged boulders of clay-slate, measuring 16 to 20 cubic feet, and embedded in muddy sandstone, indicate the transporting agency of ice.

The faults appear to run along the strike of the Barakur beds, judging from those already met with. The greatest dislocation is about 100 feet, but they do not extend for any great distance, and rapidly tail out from the point of greatest disturbance The circumstance which has preserved the field from denudation, has, without doubt, subjected it to the faulting influences of lateral pressure, and for that reason much faulting may be expected to have occurred over the whole area of the Coal-measures.

No igneous rocks have been observed breaking through the Gondwana deposits, but on the western edge of the field there is a local outburst of amygdaloidal trap. It forms a small hill on the line of the boundary-fault.

The seams of coal, in descending order are:—The thick coal-seam (Fig. 2, Plate XII.) which has been proved, at various points by borings, to vary from 30 to nearly 50 feet in thickness, and consists of alternating beds of coal and carbonaceous shale, but has not been thoroughly tested by actual working. A superficial trial-working was made near an out-crop and a quantity of fairly good coal selected from the part explored. This sample was tried under stationary boilers and found to steam well. It assays:—

Fixed carbon	52·5
Volatile matter	34·5
Ash	13·0

It is highly probable that in deeper workings it may be found to contain useful beds of coal, associated (as in most thick seams) with beds of shaly matter.

The second seam, which lies about 100 feet below the thick coal, is very variable in thickness, ranging from 2 to 5 feet. It is a shaly coal mixed with bands of carbonaceous shale; and after a considerable trial down from the outcrop it was abandoned as worthless. The best portions of this seam contain 20 per cent. of ash.

The third seam lies about 35 feet below the second, and is also of very variable thickness, ranging from 2 to 9 feet. It is a hard, strong coal, containing 30 per cent. of ash, and is therefore worthless for commercial purposes.

The fourth or King seam, so called from its discoverer Dr. King, is the seam of greatest commercial interest at the present time, and is being extensively worked from the outcrop. It is a hard, semi-bituminous, steam coal, varying from 3 to 7 feet in thickness, and generally uniform in quality. The specific gravity is 1·34, and it assays:—

Fixed carbon	58
Volatile matter	32
Ash	10

Two analyses by Mr. Tookey, F.C.S., are as follows:—

Carbon	66·45	Volatile matter	25·25
Hydrogen	4·19	Coke—	
Oxygen and nitrogen ...	10·61	Carbon ... ... 56·50	
Sulphur	0·45	Ash ... ... 10·65	
Water	7·60	———	67·15
Ash...	10·70	Water	7·60
	100·00		

It yields a gas of fair illuminating power, but not a good coke, as the coal retains its original form when burned in a retort.

It forms a useful steam coal, burns freely, leaving a buff-coloured powdery ash, and does not clinker on the firebars. Exhaustive trials have shown it capable of evaporating in locomotive boilers over 7 lbs. of water per 1 lb. of coal. It is slightly inferior to that from the Bengal coal-fields, but superior to the Warora coal, and compares favourably with some Australian coals, as shown by Mr. Price Williams in his comparative tests (Appendix A).

Extensive trials on the Madras railway by Mr. Phipps have shown it to be from 10 to 12 per cent. inferior to Bengal coal, but at the same time it is a valuable fuel for steam-producing purposes. The coal does not stand exposure to the weather, for any length of time, without crumbling and

deteriorating to a great extent. If screened and carefully stacked, it will, however, stand for a couple of months fairly well. When stacked in bulk it is very liable to spontaneous combustion, if laid thicker than 5 or 6 feet. This is not due to pyrites, of which it contains very little, but to its affinity for oxygen which sets up a rapid development of heat sufficient to ignite the mass. The only safeguard is to stack thin and so allow the heat to escape more readily. Stock heaps were generally tested with iron jumpers forced down through them, and the point of greatest heat-development was usually found about 2 feet below the surface.

The King seam, extending over 9 square miles, and assuming it to have an average thickness of 5 feet, should contain 47,520,000 tons of coal, which, after making the most liberal allowance for loss and unworkable coal, should afford ample supplies of fuel without drawing upon what may be found of value in the other seams.

Method of Working.

The whole coal-field (Fig. 1, Plate XII.) lends itself to easy and economical working; its seams are found at shallow depths, commencing at the outcrops with slight coverings of the Kamthis strata, and dipping at low angles to the centre of the basin, where the greatest depth probably does not exceed 800 feet. The measures contain very little water, and when won by proper sinkings will readily drain to the pits and give very little trouble or anxiety.

At present the King seam has been opened out by means of incline-drifts through the Kamthis rocks, driven at a gradient of 1 in 5 to the outcrop in the direction of the dip of the seam. A shaft for ventilating purposes is also sunk to the coal in each case. Fig. 3, Plate XII., shows the outcrop of the seams and the overlying Kamthis measures, with a drift going down through the latter, and illustrates the present system of opening out.

The incline is continued along the dip of the seam, and workings opened on either side on the bord-and-pillar system. The levels are driven right and left at an angle of 95 degs. from the course of the incline, giving them a slight fall out-bye to facilitate tramming from the faces to the incline hanging-on, where a double-way is laid into each tramway-level.

In the early days of the colliery, the demand for coal in large quantities necessitated rather extraordinary measures being taken to get it. All the workings were made 12 feet wide and the pillars 12 feet square, thus removing about 75 per cent. of the seam in the first working. Fortunately the roof is a model one, and the seam is very strong, or a collapse might have occurred. It was, however, evident that this could not be continued,

the pillars were increased as the depth became greater, and the levels reduced from 12 to 9 feet in width. Under 150 feet of cover the pillars are made 24 by 21 feet or 30 by 15 feet, thus removing about 54 per cent. of the coal in the first working, without in any degree causing symptoms of crushing. The remaining coal of the pillars will be easily got by the use of chock-timbers, after a number of skilled miners have been trained.

Broken working has not been attempted on a large scale, as the coolies are naturally timid, working as they are under new conditions; but a small area has been worked in the broken, and only 10 per cent. of the coal lost under a cover of 120 feet.

Fortunately, in driving these workings to the deep, no great quantity of water has been encountered: otherwise very slow progress would have been made. As it is, the dip faces are always more or less wet and necessitate the employment of a large number of water-bailers to carry the water out to the waterways above, which conduct it to collecting-sumps, whence it is pumped to the surface by means of duplex ram-pumps driven by compressed air.

There are six mines at present worked by means of incline-drifts, as above described, and one small shaft from which coals are being drawn.

Naked lights are in general use, and consist of small tin-lamps of tea-pot form, capable of holding about 6 ozs. of a mixtore of $\frac{2}{3}$ of castor oil, and $\frac{1}{3}$ of kerosene. They are provided by the company.

The rate of progress made in driving the headings is exceedingly slow, when judged from an English point of view, and renders very tedious the work of driving out a sufficient number of working-faces to ensure a fair output. About 35 to 40 feet of 12 feet wide drifting is all that can be expected from six men working double in 8 hours shifts for a month's work, or about 12 cwts. per man per shift. This is more than the average output per man, which is about 10 cwts. per day per hewer.

The hewer undercuts the seam to a depth of 3 feet, and nicks one side of his working to a similar depth; and after drilling the shot-holes with a jumper the shot-firer charges and fires them. The hewer afterwards throws back the coal and trims his place ready for another driving. The shots are put into thin cotton-bags, into which the fuze is tied. The consumption of powder amounts to a little over 3 ozs. per ton of coal wrought. English powder is used, as the local supply is not satisfactory.

The loading into tubs is done chiefly by women and boys, who carry it in baskets from those faces into which the tramways cannot be got. Each woman will load about 1 ton in the shift of 8 hours.

Any timbering that may be required is done by a rough carpenter, and one or two coolies assisting. They cut the timber into the required lengths upon the surface, and take it where required.

Platelaying is done by a special man with assistant coolies; he lays any new line required, and maintains all the roads.

Tubs are trammed from the loading-faces by gangs of trammers, who make up the engine-loads, hang on the ropes, and signal the engineman.

Each shift is in charge of a native deputy or *mukadam*, who is responsible for the proper carrying on of the work and the due attendance of the people in his shift.

The daily superintendence of each mine is carried out by a native or Eurasian underlooker of some education and standing among the people. He is in attendance for the greater part of the working hours during which coal is drawn, namely, from 6 a.m. to 10 p.m., or 16 hours in all. He has charge of the mine under his superior, and sees that the workings are kept in a safe and proper condition by frequent inspections, he also sees that all the working-faces are kept true to centre-lines.

Three European assistants daily supervise the underground workings, controlling and directing all labour under them, and giving the course of all levels by centre-lines. They daily consult with the manager and report the condition of their division.

Owing to the roof of the seam being of a strong sandstone, and the general care exercised in superintendence, accidents have been very rare, considering the inexperienced labour employed.

In 1890, there was an average of 1,845 people employed, and the accidents were as follows:—

Accidents	Underground.	Surface.	Total.
Fatal	2 ...	1 ...	3
Serious	6 ...	7 ...	13
Slight	16 ...	5 ...	21
	—	—	—
	24	13	37

In 1891 there were 2,686 people employed, and the accidents were as under:—

Accidents.	Underground.	Surface.	Total.
Fatal	5 ...	2 ...	7
Serious	9 ...	11 ...	20
Slight	2 ...	7 ...	9
	—	—	—
	16	20	36

Experience in Indian coal-mining has shown that, owing to the small production per man, it is very difficult to work up a large output with

any degree of rapidity. It is on that account usual in shallow workings to remove from 60 to 70 per cent. of the coal in first workings, leaving very small pillars, and thus more rapidly win out an increased number of working-faces. As the cover increases, the size of pillars has of necessity to be increased, and the rate of progress becomes slower. To procure an output of 500 tons per day, about 1,000 coal-cutters are required and not fewer than 200 working-places. Experience has also shown that, for rapidly developing large outputs of coal in India in shallow seams, the mines must be numerous. In deep mines, rapid development and large outputs from individual mines is almost impossible with such labour as is available, if the due safety of the mines be considered.

When the time comes to work the deeper coal in India, suitable coal-cutting machinery will no doubt have to be employed. Shaft pillars from 100 to 200 feet square would occupy from 8 to 10 months in forming, and the number of working-faces that could be started in pillar workings would depend upon the depth. Allowing for a depth of 300 feet, and shaft pillars 200 feet square, the pillars should not be of less size than 30 by 60 feet, and that would allow of about 32 working-faces being started outside the shaft pillars; these if kept at work by a full complement of 192 men, would barely yield 100 tons per day. An output of 300 tons a day from one pit would be good work in India, and for that no engine running at a high speed would be required, as the drawing probably would be distributed over the greater part of the 24 hours.

The inefficiency and lethargy of the Indian labourer render it impracticable to get large quantities in a few hours as in England. To the Eastern mind time is no object.

The tramway-gauge is 2 feet, and is laid underground with 18 lbs. steel rails and steel sleepers; on the surface the rails are of 22 lbs. section, and laid on Grigg sleepers. The coal-tubs are built of $\frac{1}{8}$ inch steel plates (fitted with steel axles and Hadfield steel wheels 12 inches in diameter), and are of the following dimensions:—Length, 4 feet 4 inches; breadth, 2 feet 8 inches; and depth, 2 feet 3 inches. The tubs each carry about 12 cwts. of coal. A small number of tubs are built 6 inches lower in height, to accommodate lower portions of the seam. The first consignment of tubs was procured from England, but they are all now built at the colliery.

The whole of the mines are connected with a central heapstead by surface tramways, by which the coal is transported for loading into railway trucks after being screened. The heapstead consists of an earthwork embankment, faced by a masonry retaining-wall 25 feet high.

There are two ordinary bar-screens, which separate the coal into two classes: the best or steam coal being that which passes over $\frac{5}{8}$ inch bars having $\frac{3}{4}$ inch spaces between them. The small coal which passes through is afterwards screened by hand into nuts, over a wire-net screen of $\frac{3}{8}$ inch square mesh, as the market demand requires. At present a very large proportion is thrown upon the slack heap, there being no great demand for this class of coal, although it is excellent for raising steam in stationary engines. The proportion of steam coal produced is about 72 per cent. of the output. The remainder consists of about 16 per cent. of nuts and 12 per cent of dust coal.

The coal is hauled from the mines to the heapsteads by small locomotive engines of the following dimensions:—Cylinder, 7 inches in diameter and stroke of 12 inches; driving-wheel, 2 feet in diameter; the boiler, fitted with 51 tubes 2 inches in diameter, works at a pressure of 140 lbs. per square inch. They are practically able to haul a load of 14 tons up a gradient of 1 in 45, and including delays in shunting and making up the sets will transport a load of 20 tons of coal 1 mile per hour, including running back with the empty tubs on an average gradient of about 1 in 80 falling with the load.

No. 1 *Incline.*—The hauling engine has two cylinders, each 16 inches in diameter and 2 feet stroke, geared $3\frac{1}{2}$ to 1. There are a pair of drums with clutch-gearing, each 4 feet $1\frac{1}{2}$ inches in diameter by 18 inches wide, the depth of flange being 1 foot $1\frac{1}{2}$ inches. Steam is supplied at a pressure of 60 lbs. per square inch from two Lancashire boilers situated close by. There are four boilers, each 30 feet long by 7 feet in diameter, but only two of them are worked at once. The boilers also drive the workshop engine, which has a cylinder 12 inches in diameter and 2 feet stroke, and the air-compressor engines, which work alternately.

The No. 1 air-compressor has two cylinders, each 18 inches in diameter and 32 inches stroke. The air-cylinders are of the same dimensions. The No. 2 air-compressor has one cylinder 14 inches in diameter and 2 feet stroke: the air-cylinder being of the same dimensions. The No. 3 air-compressor has two cylinders, each 9 inches in diameter and 18 inches stroke. These air-compressors were originally erected to drive two Stanley coal-cutting machines, and facilitate the winning out of working-faces by hastening the driving of the main galleries. The coal was found to be too hard for the machines to work effectively, and as they were constantly breaking, their use was abandoned. The air-compressors are now used to drive the underground pumps and a small underground hauling engine. The air is conveyed to the various pumps in wrought-iron

pipes with screwed socket-joints, which are found to stand the changes of temperature best. There are six small direct-acting pumps worked by this means as required, and the most distant is over a mile from the air-compressor. The use of compressed air is highly suitable in Indian mines where, owing to the general high temperature prevailing, steam is not generally admissible, and the exhaust air assists in cooling and ventilating the workings.

No. 2 Incline.—The hauling engine has two cylinders, each 8 inches in diameter and 10 inches stroke, geared $4\frac{1}{2}$ to 1. There are two drums, each $2\frac{2}{3}$ feet in diameter, $10\frac{1}{2}$ inches wide, and 10 inches deep, with clutch-gearing. Steam is supplied at a pressure of 80 lbs. per square inch by a Cochrane boiler, $5\frac{1}{2}$ feet in diameter and 11 feet high.

No. 3 Incline.—The hauling engine has two cylinders, each 12 inches in diameter and 12 inches stroke, geared $4\frac{1}{2}$ to 1. The drums are fitted with clutch-gearing, and are 4 feet in diameter, 12 inches wide, with 1 foot flanges. Steam is supplied at a pressure of 80 lbs. per square inch by a vertical Cochrane boiler, $5\frac{1}{2}$ feet in diameter and 12 feet high.

No. 4 Incline.—The hauling engine has two cylinders, each 14 inches in diameter and 18 inches stroke, geared 4 to 1. The drums are fitted with clutch-gearing, and are $4\frac{1}{2}$ feet in diameter, with flanges 1 foot deep.

No. 5 Incline.—The hauling engine has two cylinders, each 8 inches in diameter and 13 inches stroke, geared 5 to 1. The drums are 3 feet in diameter, 9 inches wide, with 9 inches flanges. Steam is supplied at a pressure of 80 lbs. per square inch from a vertical Cochrane boiler, $5\frac{1}{2}$ feet in diameter and 11 feet high.

No. 6 Incline.—The hauling engine has two cylinders, each 14 inches in diameter and 2 feet stroke, geared 4 to 1. The drums are 5 feet in diameter, $1\frac{1}{2}$ feet wide, with flanges $10\frac{1}{2}$ inches deep, and fitted with clutch-gearing. Steam is supplied from a Cochrane boiler, $6\frac{1}{2}$ feet in diameter and 14 feet high, which will work up to a pressure of 100 lbs. per square inch.

Shaft.—A small shaft which is 100 feet deep is fitted up with a light iron-headgear and a single cage, in rope guides, with a counterbalance-weight. The winding engine has two cylinders, each 7 inches in diameter and 10 inches stroke, geared $9\frac{1}{2}$ to 1. The drum is 53 inches in diameter and 8 feet wide. An old ship winch is also fixed on the surface, and a rope taken down the shaft to haul coal from workings to the deep of the pit. The ropes in use are Lang and Elliot locked coil of different sizes. The latter make of rope has given great satisfaction, and is being generally adopted.

The pumps in general use are small quadruple duplex rams, and simple direct-acting force pumps, with from 6 to 12 inches steam cylinder and 3½ to 6 inches rams and plungers. Compressed air is the motive power underground, and wrought-iron pipes with screwed joints are in general use.

The workshops are built of iron framing covered with corrugated galvanized sheets. The objection to the use of this material in a hot country like India is that during the hot season the temperature inside often rises to 120 degs. Fahr., a heat too great for even natives to work in well. They are fitted up with every class of machine that may be required for colliery work, including screw-cutting and pattern lathes, shaping and slotting machines, drilling machines, punching and shearing machines, bolt and screw-cutters, plate-bender, emery wheels, grindstone, Root blower for Smith's forges, band-saw, circular saw-bench, 7 cwts. steam hammer, and all necessary workshop tools. With the foregoing equipment all classes of repairs and constructive work are executed with facility, and all small crucible castings of iron and brass are made at the colliery. The workshops, engines, and machinery are in charge of an English mechanic, assisted by an Eurasian foreman. Many of the native artizans have become excellent workmen, and, although not such vigorous workers, are capable of turning out quite as good work as our own artizans.

Labour.

The supply of labour from local sources is precarious. Mining being new and strange to the people, and land being plentiful and cheap with a sparse population, the local people can only be depended upon in the months during which they are not occupied in cultivation. They have to some extent got over their timidity, but the class attracted by mining are from the very lowest castes of the population, and are by no means desirable as workmen.

The Telingana cooly is a shiftless creature, a slave bred by centuries of oppression from his former rulers, the Hindoos and Mahomedans. The Telingee measures the comfort of existence by the quantity of food and liquor he can get. They are a most improvident race, living to be in debt and drinking as long as they possess any money. It was a difficult matter to get them together after the pay so long as the money lasted, and often from 40 to 50 per cent. were absent for three or four days at this time. The castes represented are the Koombis or cultivators, Goola or shepherds, Dhirs, shoemakers, and Pariahs, the last three being the lowest and most numerous of all. The inhabitants of the

jungle villages have not taken well to underground work, but come in for surface labour. They are excellent woodmen, and for cutting and dressing timber are difficult to excel.

In contrast to the people of Telingana are the men from Northern India and the Maratta country, who are industrious and frugal to a degree, and can always be relied upon as regular workmen. These men return to their country after saving a little money, and frequently make use of the post office to remit to their families at home who are settled on the land as cultivators. Very few men are accompanied by their wives and families, and the attachments formed are generally of a temporary and unprolific nature, so that the juvenile population is very small, and the time when a race of men will be bred to mining on the spot is very far off.

A few Mahomedans are employed, but they do not as a rule take to labouring work, and are chiefly employed in official capacities or as artizans and engine-drivers.

The early experience of the management impelled them to adopt strict sanitary measures for ensuring the health of the place, which is situated in the midst of a wide jungle. The malarial emanations cause numerous fevers in the cool season, and the local water-supply being both meagre and bad in quality, combined with the general filthy habits of the people, if left to themselves, necessitated strict attention to sanitary matters.

Several blocks of stone buildings with galvanized iron roofing were built for the occupation of the imported people. The quarters were made about 10 by 12 feet each, and are quite large enough for three or four natives to occupy.

Lines were set out in squares for those who erected their own huts, care being taken that each was separated from its neighbour so as to facilitate sweeping up of the refuse, and afford isolation in case of fire. A hospital was erected, and a medical and sanitary staff organized, consisting of the medical officer, hospital assistant, rangers, and sweepers.

The rangers are on daily duty to prevent the fouling of the lines and compel the use of latrines, while the sweepers clean up all the dirt and filth that daily gathers about native quarters. They also visit the underground workings daily, and remove all excrement deposited there. The people are moreover prevented from drinking any filthy river water, which they prefer, if it be only from a running stream.

It soon became evident that the local well water was inadequate for the supply of a large population, and some springs of water situated about 3 miles from the colliery were taken advantage of. They issue from some limestone-beds of the Vindyan formation, in the bed of a river,

and have never been known to run dry. After the rainy season the quantity issuing is about 100 gallons a minute, but towards the end of the dry season it has been measured as low as 20 gallons per minute. A line of cast-iron pipes, 6 inches in diameter, was laid, and a small steam pump and boiler erected at the springs. The water is pumped to the colliery, and distributed to the different mines.

At present, the underground feeders, though small, are sufficient for boiler purposes, but owing to mineral and sewerage impurities are unsuitable for human consumption.

The result of the careful sanitation and superintendence of the water-supply has been an almost total absence of infectious epidemics during the past three years. Only one slight outbreak of cholera occurred from an imported case, and that was quickly stamped out. Careful sanitation and care of the water-supply from contamination will give almost complete immunity from cholera, that dread scourge of Indian towns. It is a very common sight to see natives perform their ablutions in the same stream from which they take their drinking water, and yet they are most particular in having their water-vessels clean and untouched by lower caste people. They prefer to dip their vessels into the well or pool, and some difficulty was experienced in getting them to use pumps. They got over this by each person or caste pumping their own water.

The majority of the people build their own houses in a very simple manner. With a few poles and grass-thatching they form a covered space 9 or 10 feet long, and 6 or 7 feet broad, 6 feet high, and plaster the floor and sides, which are only 2 feet high, with mud and cow dung. These do not cost more than 3 or 4 rupees each, timber being provided by the company. Rooms about 12 feet square, built of stone in blocks with corrugated roofing, cost about 80 rupees each. This roofing is not suitable, as it becomes excessively hot during the hot season. A very cheap and efficient house can be built with mud walls and thatched roof. The mud once dried will stand a very long time and becomes almost as hard as stone. Houses for European officials require to be solidly built to ward off the heat of the sun, and cost about the same (if built well) as similar buildings in England. A very good house of three rooms, one, say 20 by 30 feet by 18 feet, and two 20 by 20 by 15 feet with verandah and servants' quarters, can be built for about 8,000 rupees or £500.

At Singareni the men are paid monthly, two or three days being usually allowed to prepare the pay bills after the last day of the month. Each man is paid separately, with the exception of the coal-cutters, who are paid in gangs of four or six, as they work together on the same work.

The workshop foreman has consolidated pay of 150 Halli Sicca rupees per month with no fixed hours; two or three of the superior native fitters are also under consolidated pay of from 60 to 90 Halli Sicca rupees per month. These men are quite as good as the average European fitter. The mechanical department of the colliery is under the superintendence of a good English mechanic who has gained his mining experience there. No home-trained colliery enginewright is on the staff. The mechanical and artizan classes employed upon the colliery are worthy of every commendation. The capability of improvement shown by them, and their present efficiency are remarkable. The writer has not seen better work turned out of any colliery workshop at home, and the condition in which they keep the engines and pumps is equally satisfactory.

The underground labour is all divided into 8 hours shifts, and is paid at slightly higher rates than for surface-work. Coal-cutters are paid by the yard. The standard price being 5 Halli Sicca rupees per yard for a heading 12 feet wide and 5 to 7 feet high, places that are narrower or lower are paid at proportional rates. The earnings per man vary considerably, 10 rupees per month being about a minimum, and 20 rupees about the maximum earnings, all depending upon the efficiency of the workmen. Broken workings are paid at about two-thirds the price of whole workings.

Payments are witnessed by the underlookers and European officials, and are made in the local currency of the State of Hyderabad known as Halli Sicca rupees; the smaller coinage consists of dubs or pice, which are irregular pieces of bronze stamped with Persian characters. A more barbarous currency than the dubs could not well be imagined. The Halli Sicca rupee is of inferior value to that of the British Government, the nominal ratio being as 120 to 100, but the market value varies considerably, the average exchange rate being about 116. The Halli Sicca rupee is nominally equal to 96 dubs, six of which equal 1 anna, and one is equal to 2 pies. The market value of these varies from 94 to 104 per rupee, according to their scarcity or abundance in the bazaar.

The rates of wages paid to different classes of labour are as follows :— Surface coolies, who work from 6 a.m to 6 p.m., with two hours' interval, are paid 5 annas per day; women get 3 annas per day; heapstead men, 7 annas for 8 hours; fitters, 8 annas to 3 rupees for 10 hours; smiths, 8 annas to 1 rupee 8 annas for 10 hours; smiths' apprentices, 4 to 8 annas for 10 hours; carpenters, 10 to 14 annas for 10 hours; enginemen, 8 to 12 annas for 8 hours; firemen, 6 to 7 annas for 8 hours; and locomotive drivers, 12 annas for 8 hours. Fillers are paid 6 annas per day, when not paid by contract; women fillers get 3½ annas per day;

children, 2 annas; trammers are paid 7 annas, and general coolies 5 to 6 annas; deputies or mukadams are paid 16 rupees per month; underlookers, 30 rupees per month; head platelayers, 12 annas per shift; carpenters, 12 annas; banksmen, at mouth of inclines, 6 annas; timekeepers, 20 to 25 rupees per month. Native clerks are paid 30 Government rupees per month, and a native chief accountant and cashier, 120 Government rupees per month.

THE NUMBER OF NATIVE WORKMEN EMPLOYED IN JULY 1892, WERE AS FOLLOWS:—

	Men.	Women.	Children.	Total.
Surface —				
Mechanics	19	—	—	19
Engine and pump men and firemen	91	—	3	94
Apprentices	5	—	—	5
Carpenters	13	—	—	13
Smiths and pick-sharpers	22	—	12	34
Labourers	13	2	—	15
Trammers	57	—	—	57
Screeners	12	—	—	12
Shunters	35	—	2	37
Stacking slack	24	60	13	97
Tramway men	30	—	—	30
	321	62	30	413
Underground—				
Coal-cutters	1,346	—	—	1,346
Fillers	141	357	56	554
Trammers	66	—	—	66
Platelayers	25	—	3	28
Watermen	94	—	3	97
Stonemen and shifters ...	138	38	3	179
Timberers	21	—	—	21
Deputies	38	4	—	42
Ventilating	7	—	7	14
Boys (greasers)	—	—	8	8
Miscellaneous	93	5	1	99
Petty contractors (coolies)	74	43	1	118
	2,043	447	82	2,572
Totals	2,364	509	112	2,985

MARKETS.

The chief consumers of the coal at present are as follows:—

Railway.	Quantity per Annum. Tons.	Price per Ton at Colliery. Rupees.
Great Indian Peninsula	50,000	... 6
Nizam's Guaranteed State ...	16,000	... 6
Madras (sold through a contractor).	18,500	... 5
Southern Maratta	15,000	... 6¼

There are also a few mills and small works in Hyderabad and neighbourhood which consume nut coal.

The great distance from large centres of consumption and the consequent high cost of carriage militate against a large sale being made. The railway's minimum charge at present is 4½ pies per ton per mile, which amounts to 2·34 rupees per 100 miles, and even at this comparatively low freight the coal cannot be put on distant markets to compete with English and other imported fuels. The opening of the new east coast railway to Madras will no doubt enable it to be got into that district at a lower rate, but even with the cheap water-carriage available from Bezwada, that has been found difficult owing to the low rate at which Bengal coal can be brought in by sea.

The delivery of Singareni coal in better condition by direct railway carriage will no doubt assist its introduction, and when some further extensions of the Southern Maratta railway system are completed the approach to the Kolar gold-fields will be shortened and it may be got there at a fair price.

There is very small hope of ever opening out an export trade from the coast, as the coal would require to be loaded into lighters and taken from six to eight miles out to sea to be got on shipboard. Widely extended sales will depend upon low freights and reduced selling price at the colliery.

With the exception of the Madras railway, those above mentioned are supplied direct by the mining company; all other consumers purchase through a native merchant, who has contracted to take 42,000 tons of steam coal and 21,000 tons of nuts at 5 and 2½ rupees per ton respectively during 1892, and 66,000 tons of steam coal and 33,000 tons of nut coal during 1893 at the same figures. He is experiencing great difficulty in disposing of even the smaller quantities to be taken in 1892, and no market is at present evident for the larger quantities to be taken in 1893.

The capital expenditure upon the colliery, up to the end of 1891, amounted to 1,308,453 rupees, divided over the following general heads of expenditure:—

	Rupees.	As.	Ps.		£	s.	d.
Preliminary expenses	28,441	5	8	=	1,955	6	10
Construction of mines	158,750	6	1		10,914	1	9
Machinery tools and plant ...	691,947	9	6		47,571	7	11
Permanent way	113,202	5	4		7,782	13	3
Buildings	175,555	8	3		12,069	8	9
Water supply	8,751	10	0		601	13	5
Establishment	74,092	12	3		5,093	17	7
Medical and sanitation	13,705	6	2		942	4	11
Survey...	15,920	10	1		1,094	10	10
General contingencies	28,085	10	5		1,930	17	9
	1,308,453	3	9		89,936	3	0

For the amount of work done, the sum expended on preliminary expenses and construction of mines and buildings is exceptionally high, chiefly due to a forced development under unqualified and amateur supervision in the early days of the colliery. A large sum of money was also expended upon needless machinery, such as a Schiele fan and engine, iron head-gear, and winding-engine, all very good plant but bought before they were wanted. The cost of opening the early mines was about double what it ought to have been, and the buildings cost extravagant prices for badly finished work.

The progress of the output is shown by the following figures:—

	Tons.
1887	3,258
1888	13,382
1889	59,646
1890	125,470
1891	144,688
1892 (half-year)	75,875*

Previous to 1890 the cost accounts were not kept with any degree of care, but in that year the output cost, exclusive of royalty, 2 rs. 9 as. 4·88 ps. per ton; in 1891 it cost 2 rs. 4 as. 11·72 ps. per ton; and for the half-year ending June, 1892, it was 2 rs. 6 as. 3·15 ps., showing a reduction on the cost of 1890 as per detailed statement below:—

1890.	Rs.	As.	Ps.
Underground labour	1	6	10·55
Aboveground labour	0	6	7·03
Materials	0	7	4·67
Charges	0	4	6·63
	2	9	4·88
Rent	0	7	5·09
Total cost per ton	3	0	9·97

1891.	Rs.	As.	Ps.
Underground labour	1	4	11·93
Aboveground labour	0	5	9·40
Materials	0	5	9·45
Charges	0	4	4·94
	2	4	11·72
Rent	0	7	3·49
Total cost per ton	2	12	3·21

1892—Half-year ending June 30th.	Rs.	As.	Ps.
Underground labour	1	5	1·83
Aboveground labour	0	6	11·29
Materials	0	6	1·24
Charges	0	4	0·79
	2	6	3·15
Rent	0	7	0·39
Total cost per ton	2	13	3·54

* The last three years show the most rapid development of output in the history of Indian coal-mining.

The charge for rent has been fixed on the basis of 22 per cent of the nett profits, taking a fixed working cost of 2 rupees 6 annas in Government currency.

These figures in the early days of the colliery promise well for future cheap production, after labour becomes settled and trained to mining.

At present the slack and a large quantity of nut coal are unsalable. In 1890, this amounted to 24,312 tons, which was stacked, and destroyed by spontaneous combustion. The sales in that year amounted to 101,158 tons, which realized the average rate of 5 rs. 9 as. 8·7 ps. per ton, and the cost per ton sold, allowing for royalty, was 3 rs. 12 as. 6·8 ps., leaving a balance of profit of 1 r. 13 as. 1·9 ps. per ton.

In 1891, 27,335 tons of unsalable nuts and slack were produced. The sales amounted to 117,353 tons, which realized an average price of 5 rs. 7 as. 8·8 ps. The cost per ton sold, including royalty, amounted to 3 rs. 6 as. 7 ps. per ton, leaving a balance of profit of 2 rs. 1 a. 1·8 p. per ton.

In 1892, the contracted sales amount to 149,000 tons, which, allowing for unsalable slack, will require an output of 176,000 tons. The average selling price is 5 rs. 2 as. 7·87 ps., with an estimated profit of 1 r. 14 as. 7·8 ps.

The prospective sales for 1893 amount to 185,000 tons, which will require an output of 210,000 tons, leaving a balance of unsalable slack amounting to 35,000 tons to be thrown aside. The average selling price will be 4 rs. 15 as. 6 ps., and the estimated profit, 1 r. 14 as. 7·8 ps.

The commercial progress of the colliery is shown by the following statement :—

Year.	Sales. Tons.	Profit. Nett. Rs.	Per Ton. Rs.	As.	Ps.
1890...	101,158 ...	184,384 ...	1	13	1·9
1891...	117,353 ...	243,147 ...	2	1	1·8
Half-year ending June 30th, 1892	57,406 ...	96,511 ...	1	10	10·8

The present mines will not satisfy the requirements for 1893, and it is doubtful, even if they should, whether the contractor will be able to dispose of the largely increased quantity. It must be said that the foregoing statement of profits is a handsome return upon the capital expended, and shows the profitable nature of Indian coal-mining under favourable circumstances. Unfortunately the nominal capital of the company stands at £1,150,000, with a loss already of six years' interest to the shareholders, which will require enormous sales and a very high profit to be made, as the other departments of the company's business are in anything but a flourishing condition.

APPENDIX A.—RESULTS OF TESTS OF SINGARENI COAL BY A CALORIMETER, AND ALSO ON ONE OF THE NIZAM'S GOVERNMENT STATE RAILWAY LOCOMOTIVES, No. 45 (A CLASS), IN COMPARISON WITH SIMILAR TESTS MADE OF COAL FROM VARIOUS COLLIERIES IN NEW SOUTH WALES, ON ONE OF THE GOVERNMENT LOCOMOTIVES (No. 101) OF THAT COLONY, BY MR. R. PRICE-WILLIAMS.

Description of Tests.	Date of Trial.	Coal.			Water.		Percentage of Locomotive Evaporative Power to that given by Calorimeter.	Ash.	Load on Journey.	Time.			Average Speed while Running.	Average Boiler Pressure.
		Specific Gravity.	Total Amount Burnt.	Lbs. Burnt per Sq. Ft. of Grate per Hour.	Burnt per Mile.	Water at 212° Fahr. converted into steam by 1 lb. of coal by Calorimeter.	Total Amount Evaporated.	Evaporated per Lb. of Coal.				Running.	Shunting or Standing.	Total Time on Journey.		
SINGARENI COAL.			T. C. Q. L.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	%	%	T. C.	H. M.	H. M.	H. M.	Miles per Hour.	Lbs. per Sq. In.
No. 1, goods train—Secunderabad to Gangarwaram	Mar. 23, 1892	1·30	1 8 3 19	29·87	74·03	12·42	22,712	*7·01	56·44	10·60	673 11	3 29	1 2	4 31	14·38	135
No. 2, goods train—Secunderabad to Gangarwaram	Mar. 25, 1892	1·30	1 10 0 2	24·01	76·84	12·42	23,529	*6·99	56·28	10·60	667 16	3 29	2 21	5 50	13·92	135
No. 3, goods train—Wangapalli to Secunderabad	Mar. 29, 1892	1·30	1 4 2 3	31·35	75·77	12·42	20,579	*7·49	60·30	10·60	551 12	2 53	0 46	3 39	12·94	137
No. 4, mail train—Secunderabad to Gangarwaram, *via* Hyderabad	April 3, 1892	1·30	0 15 0 1	24·85	33·12	12·42	13,027	*7·75	62·39	10·60	229 9	2 18	0 31	2 49	22·06	135
NEW SOUTH WALES COAL.																
Great Northern Colliery Company	Mar. 20, 1890	1·34	2 19 0 0	33·7	47·2	12·03	44,965	*6·80	56·5	13·60	145 0	7 53	6 0	13 53	17·50	130
Metropolitan Coal Company	Mar. 22, 1890	1·38	2 12 1 14	33·9	41·9	12·29	44,285	*7·55	61·4	13·30	148 10	6 27	5 53	12 20	21·70	135
Northumberland Coal Company, Fassifern	Mar. 31, 1890	1·42	3 3 2 3	32·6	50·9	11·40	45,092½	*6·33	55·5	19·60	151 2	8 13	7 22	15 35	17·00	125
Waratah Colliery Company	April 2, 1890	1·30	2 13 2 15	32·4	42·9	13·92	44,455	*7·40	53·1	10·26	146 7	7 58	5 16	13 14	17·50	135

* The evaporative power on locomotives, according to Molesworth (p. 459), varies from 5·2 to 8 lbs.; Webb's compound locomotive has evaporated as much as 9·49 lbs. of coal.—Mr. Worthington on "Compound Locomotives," vol. xcvi., *Proceedings of the Institution of Civil Engineers*, 1889.

NOTE.—The small amount of water lost through the injector overflow has not been allowed for, owing to the difficulty of estimating it. It does not, however, amount to more than 1 per cent., and would not appreciably affect the accuracy of the results here given.

Appendix B.—Local Products.

The following table gives the best local timbers available, although there is a large variety of other timbers, which in other parts of the dominions afford good supplies in the Singareni jungles they do not attain a serviceable size :—

	Local Name.	Breaking Strength. Lbs.	Weight per Cubic Foot. Lbs.
Tectona Grandis	Teak	600 ...	38
Soymida Febrifuga	Somi	626 ...	73
Chloroxylon Swietenia	Satinwood ...	800 ...	57
Odina Wodier	Gumpani	300 ...	50
Pterocarpus Marsapium ...	Bijasal	800 ...	54
Hardwickia Binata	Eppa	940 ...	82
Acacia Arabica	Babul	880 ...	55
Terminalia Tomentosa	Nulla Muddi ...	677 ...	62
Anogeisus Latifolia	Tiramani	870 ...	60
Lagerstroemea Parvifolia ...	Chinangi	467 ...	50
Adina Cordifolia	Bandari	464 ...	40
Diaspyros Chloroxylon ...	Ilindu	— ...	—
Briedelia Retusa	Dudimuddi ...	525 ...	50
Grewia Tiliæfolia	Daman	565 ...	48
Milinsa Velutina	Baradudi	836 ...	50

The breaking strains are for sections of 1 inch square, 1 foot between supports, and loaded in the centre.

The prices charged by H.H. the Nizam's Forest Department are as follows :—

Teak logs—

Dimensions. Cubic Feet.	Price per Cubic Foot. Halli Sicca Rs. As.
3 to 8	... 1 0
8 to 10	... 1 2
10 to 12	... 1 4
12 to 20	... 1 6

Satinwood at same rates. Other timbers :—

Dimensions. Cubic Feet.	Cost per Cubic Foot. Annas.
3 to 8	... 8
8 to 10	... 9
10 to 12	... 10
12 to 20	... 11

Small rafters—

Girth. Inches.	Length. Feet.	Cost per 100 Feet. Halli Sicca Rupees.
18	12 to 15 ...	... 25
15	12 to 15 ...	... $12\frac{1}{2}$
12	12 to 15 ...	... $6\frac{1}{4}$

Prop-timber up to 24 inches girth is charged $\frac{1}{2}$ anna per foot of length; a 6 feet prop thus costs 3 annas, plus the cost of cutting and leading.

Owing to the comparatively light rainfall the trees seldom attain any great size. The Eppa attains full growth and is a noble tree in the Singareni jungles. The others seldom exceed a size of 30 cubic feet in the log.

The local teak wood is very small and only useful for sleepers or small work. It is beautifully grained, and when polished becomes very ornamental.

Somi wood is a very fine timber resembling mahogany, and is chiefly used for ornamental work.

Satinwood is found of fair size, and is chiefly used for ornamental work and for bushing cart wheels.

Gumpani wood is one of the few Indian timbers readily sawn by the circular saw, and is most useful for common deals, although not a very durable wood.

Bijasal wood is a very fine redwood and supplies the place of teak. It is easily worked, and may be safely applied to almost any purpose about a colliery, being very strong and durable. It possesses the peculiarity of seasoning as it stands in the jungle, after attaining its growth.

Eppa wood is an exceedingly fine timber, though the great objection to its use is that its excessive hardness makes it difficult to saw and work; but when once seasoned it is very durable, and may be relied upon to stand very heavy strains.

Babul wood is not much used, but the bark is very useful as yielding a large amount of tannin, and where the tree is found large it is a very strong and valuable timber.

Nulla Muddi wood is a strong-grained timber found in abundance. It is not suitable for planking as it is liable to split, but for roofing purposes and supports it is excellent.

Tiramani is a very abundant and useful tree, and supplies the bulk of the timber for underground use. It grows straight and of small diameter. It is also useful for tool and pick-handles, cart-shafts, etc.

Chinangi wood in some districts is in great favour, but in the Singareni district becomes frequently hollow after attaining full growth. It is tough and elastic, and very useful for axe-handles, buggy-shafts, and other similar purposes.

Bandari is a tree of rather large dimensions, which yields a yellowish wood useful for indoor work, but which will not stand outside or resist the attack of white ants.

Ilindu is a small evergreen tree which is used largely for pick and axe-handles.

Dudimuddi is a rather hard wood, which seasons well, takes a fine polish, and stands well in water.

Daman is a small tree used for pickaxe-handles, shafts, oars, etc., and is very tough and elastic.

Baradudi is a yellowish timber, also largely used for pickaxe-handles.

The timbers used for pick-handles, although inferior to American hickory, answer their purpose remarkably well, and are very cheap. Imported hickory handles cost, at the colliery, 10 rupees per dozen, while those made by hand from the local timbers cost only 2 rupees per dozen.

Great care is necessary to avoid the use of timber liable to be attacked by white ants, as these pests very quickly destroy the ordinary timbers of temperate climates and also many of the inferior jungle timbers. The best protection for European timber, if sent out in the form of engine-framing, etc., is a liberal application of coal-tar or creosote, but if left undressed it will be rapidly destroyed.

The local timbers described are the best and most useful available, although there is a further large variety not used, which in less favoured spots would be appreciated. Those mentioned, with the exception of gumpani, bandari, and baradudi, are very durable, and perfectly resist the white ants. They are much to be preferred to the larger and more quickly-grown teak from Burma, which is much deteriorated by being barked and allowed to season standing, thus losing its essential oils and reducing its durability. The ordinary circular saw is not of much use for working native timbers, owing to their great density and hardness. The best machine to deal with them is a deal frame-saw, but hand-labour employed in ordinary sawpits will cost about the same. Eppa, the hardest timber, costs 5 Halli Sicca rupees per 100 square feet to saw, and the softer sorts proportionally

lower rates. The workmen earn about 6 to 8 annas per day, equal to 6½d. and 8½d. respectively, and find their own saws.

Large deposits of crystalline magnesian limestone occur in the Vindhyan formation of the immediate neighbourhood, yielding very beautiful white, yellow, and blue marbles. Unlimited supplies of lime are available from these deposits, but the most esteemed building lime is prepared by burning what is known as "kunker," a limestone collected near the surface, and formed as a sort of concretion in the soil by climatic influences. This possesses remarkable binding power in buildings, almost equal to Portland cement. After a season's rain has fallen, the whole mass of work is so bound together that frequently the stone will break before the mortar. For plastering purposes, either inside or outside of buildings, it is unequalled, as it can be made to assume a polish almost equal to marble by the addition of a little talc. The flat-roofed houses so generally built in India are usually faced with the same description of plaster, and if well done in the first instance, resist the combined effect of heat and cold and rain.

The sandstones of the Gondwana measures yield a fine supply of building-stone, but no good brick-earth is found in the neighbourhood, and the Coal-measures here are without a single good bed of clay. Thin deposits occur, but these are intermittent and not available. All firebricks are purchased from England. A deposit of soapstone or talc occurs in the neighbourhood, which, if opened out, may yield a supply of fire-resisting material capable of being sawn into any required forms. It is used by the natives to make fire-resisting plates and dishes, and possibly useful crucibles might be made out of it.

There are also extensive deposits of magnetic oxide of iron in close proximity to the coal-field. The nearest of these is known as the Singareni iron-deposit. The ore occurs with laminæ of quartzite and contains in its natural state too large a proportion of silica to be practically useful. There are other deposits about 20 miles from the colliery in which the ore is purer; these are of great extent, but the district is rather difficult of access. This ore is smelted by the villagers with charcoal in small closed furnaces built of a loamy clay. The ore is free from phosphorus, and yields a splendid iron, of which they make their axes and various small articles of husbandry. There are, no doubt, the requisities of a fine iron industry here thrown together in close proximity, and awaiting only capital and enterprise to develop.

The PRESIDENT said Mr. Kirkup's paper contained much useful information, and would prove very valuable to the members. Perhaps the writer could translate the costs in Indian units into English currency, in terms such as they were accustomed to use.

Mr. J. P. KIRKUP said, with reference to the English value of the rupee, that this was such a constantly varying quantity at the present time that any calculations would prove delusive and incorrect.*

Mr. T. E. FORSTER said the relative cost of labour in a country like India, where it was necessary to employ an enormous number of men (one man producing only about ½ ton of coal per day), was interesting, and the

* The Indian currency is divided into rupees, annas, and pies. Twelve pies are equal to one anna, and sixteen annas are equal to one rupee. At the present rate of exchange the rupee is worth about 1s. 3d. The capital expenditure was calculated at an exchange of 1s. 5d. per rupee, as at the time of the transfer the exchange was higher.

result very satisfactory—when they compared the cost with the profits. He would like to ask the author whether any attempt had ever been made to work the seams on the longwall principle? It seemed to him that the manner of working in which they could most profitably employ native labour, where such an enormous amount of pit room was required, would be by having a longwall face, although it was very doubtful whether native labour could be trained to such work. He objected to the results of trials of Australian and Indian coal recorded, because some of the coals named in the table could hardly be considered as representative of ordinary New South Wales coal, and he asked that the members should not take the results as representative as far as that colony was concerned.

Mr. KIRKUP said that longwall working had not been attempted in India, and after three years' experience of native labour he would not like to start longwall until the workmen had had an extensive experience, and then only under favourable conditions. He (Mr. Kirkup) had no personal knowledge of Australian coal, and he gave the results of the trials by Mr. Price-Williams upon that gentleman's authority.

Mr. G. B. FORSTER moved a vote of thanks to the writer of the paper.

Mr. T. E. FORSTER seconded the vote of thanks, which was cordially adopted.

THE VEITCH-WILSON IMPROVED LAMP-PRICKER.

The SECRETARY exhibited a safety-lamp fitted with the Veitch-Wilson lamp-pricker and snuffer. It is similar to an ordinary pricker, with the addition at the top of a snuffer, as shown in Fig. 1. The relative position of the snuffer is arranged so that the pricker must be raised above the ring holding the wick-tube before the snuffer can pass over the wick. The distance between the snuffer and the pricker is regulated by the construction of the lamp, so that the wick must be raised before it can be snuffed; and by so doing a fresh piece of wick is exposed and lighted by the flame before the crust can be removed. The snuffer is made of such a width as to pass over the entire width of the wick, and removes all the crust, of which only a portion can be removed by the ordinary pricker.

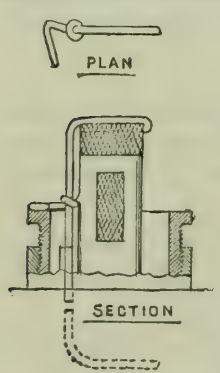


Fig. 1.

The following paper by Mr. A. L. Collins on "The Ghorband Lead-mines, Afghanistan" was taken as read :—

THE GHORBAND LEAD-MINES, AFGHANISTAN.

By A. L. COLLINS.

It is perhaps natural, considering the present condition of the country, that little is known about the mineral deposits of Afghanistan; the remoteness of the country and the unsettled and suspicious nature of its Government preventing any exploitation by outside capital, or even the investigations of the ordinary traveller. There are however several deposits which have been worked on a considerable scale to supply the local demand, almost from time immemorial; among which may specially be mentioned mines of lead, rock-salt, and precious stones. Important deposits of coal, iron, asbestos, and other minerals are also known, but are at present practically unworked.

The most considerable lead-mines in Afghanistan are situated at Frinjal, in the Ghorband valley, about 50 miles in a straight line north-west from Kabul. The country around is bare and desolate in the extreme, the valley itself being 7,000 feet above the sea-level; while both north and south the mountains of the Hindu Kush and the Paghman range reach more than double that height. The climate is however pleasant; though a good deal of snow falls in the winter, it is never deep enough to seriously interfere with mining work, while grapes and almonds ripen during the summer in the more sheltered parts of the valley. The absence at the surface of waste-tips, slag-heaps, or the usual indications of the presence of mining works makes it difficult at first sight to appreciate the real extent of the operations; it is indeed possible that the openings of many of the older excavations have collapsed, or were overlooked by the writer. But judging from the extent of the old workings known—a series of large empty chambers and narrow irregular tunnels which can be followed underground for quite half a mile into the mountain—they must certainly have been worked in spasmodic Afghan fashion for hundreds of years. They seem to have been last abandoned during the unsettled times of the latest British occupation of Kabul, and were only re-opened by His Highness the present Amir some six years ago.

The country rocks are limestones and shales, striking east-and-west in about the same direction as the valley, and dipping from 10 degs. to nearly

30 degs. northwards. The age of these rocks has not been definitely fixed, the only fossils found by the writer being a few imperfectly preserved crinoidal stems, too fragmentary for recognition; but they are probably as old as, or older than, the Carboniferous. The lead occurs in a large fissure-vein, having roughly the same strike and dip (15 degs. north) as the enclosing broad belt of limestone; the ore forms the cementing-material of a breccia composed mainly of angular limestone-fragments, varying in size from the smallest piece up to large blocks many tons in weight. (Fig. 1, Plate XIII.) Closely examined, the ore is found to be finely crystalline galena, with small patches of iron pyrites, and a little crystalline calcite, the whole being sprinkled confusedly through buff marly clay which fills all the interstices between the limestone-fragments: no galena clean enough to smelt without previous washing being found in the mine. In places a tendency towards a concretionary structure of the galena is noticed in the clay (see lower part of Fig. 1), which suggests the much more perfect little concretions of galena in the soft sandstone at Mechernich, Rhenish Prussia. It is noticeable that the larger limestone-fragments are often surrounded by a thin edge of brighter clean galena, as if they had determined the deposition of the lead from solution—just as gold in the Barberton conglomerate is often deposited as a crystalline film around the larger pebbles.* The brecciated fragments seem to be in actual contact with each other, as if they were all in position in the fissure before the advent of the lead-carrying solution; and though large surfaces of fine-grained galena and clay, up to 30 or 40 square feet in area, are often met with in the working-faces, it is only where the fragments of the breccia are themselves large, or against the hanging wall of the deposit. The vein is very wide in places, some of the workings measuring over 30 feet from wall to wall; but there is a remarkable absence of cavities, and of crystallized minerals; even secondary calcite, generally so common in such deposits, being comparatively rare. Nor is there any evidence of repeated re-opening of the original fissure; the large amounts of marly clay found on some parts of the footwall, and mixed with the galena itself, being probably due to the forces which originally caused the shattering of the limestone in the fissure, and the subsequent partial solution of the impure limestone. This idea is supported by the appearance of the limestone-fragments in the breccia, which are found on breaking to consist of unaltered hard bluish limestone in the middle, with a softer, whiter, and more decomposed envelope outside. There are many mineral springs in the immediate

* Mr. C. J. Alford, "The Geological Features of the De Kaap Gold-fields," *Mining Journal*, 1889, vol. lix., page 475.

neighbourhood, apparently on the lines of fissures, which are now depositing huge masses of travertine at the surface; the water being cold, faintly alkaline, and highly charged with carbonic acid gas. It is possible that such a spring of alkaline water originally deposited the galena in the Ghorband lode, dissolving out some of the pure carbonate of lime and leaving the insoluble clay behind; and that if the travertine caps of the existing springs could be removed it would be found that similar ore deposits are even now in course of formation.

The lode weathers at the surface to a well-defined ferruginous gossan, the lead disappearing, and the pyrites being decomposed so as to colour all the cementing material between the limestone-fragments: the brecciated nature of the whole being still distinctly visible. This gossan can be traced at the surface for over a mile, and seems to have been worked upon in several places; a few small specimens of cerusite were found in it by the writer, who was informed by the workmen at the mine that considerable amounts of that ore had at one time been treated.

The mine is worked by the Afghan Government, a colonel from the army being in charge, at a salary equivalent to about £2 10s. per month. This official has, of course, no knowledge of mining; he was, in fact, much disgusted at being required to accompany the writer underground; but judging from the style he keeps up it is obvious that he has some other means of supplementing his income. There are about 600 men kept working, half by day and half by night; and they are drawn in rotation from 1,800 men of the whole district for 30 or 40 miles around, each man serving ten days at a time. These men are all Hazaras, a race of small and active men of Mongolian type, who make cheerful and hard-working miners; a small guard of Afghan troops being, however, kept at the mines to prevent disturbances. The men agree among themselves where and how to work, each gang of ten choosing a *dah-bashi*, or foreman, to arrange such matters, and to receive and divide the pay. The ore has to be brought out of the mine, crushed, and washed clean, ready for smelting, by the miners themselves, being paid for at the rate of one Kabuli rupee for four seers, or about 35s. per ton of 2,240 lbs., out of which deductions are made for steel and oil. If fairly paid this would give something under 1d. per day to each man, as against the 3d. or 4d. which would be the proper pay, *e.g.*, of a soldier, but they are often cheated even of the balance fairly due, by the Afghan scribes and officials; so that even the "tail of the cow," as they pathetically put it, is withheld from them. As a matter of fact, all seemed to bring their food with them to the mines when they arrived for their ten days' service, putting very little faith in the possible pay.

The mine, which is quite free from water, is approached by an incline from the hillside, with a slope of about 15 degs., which seems to follow pretty closely the footwall of the vein. This soon opens out into a great chamber, over 54 feet long and 18 feet high, quite one-half of which is metalliferous cementing-material, and which at present yields nearly one-half of the ore obtained. Other workings ramify, up or down, in every direction from the chamber. All work is done with hammer and gad—a short-handled hammer of about $2\frac{1}{2}$ lbs. weight, often little more than a cubical or somewhat oblong lump of iron, with a hole down the middle for the hilt, and a steel gad of from 5 to 10 inches in length, being commonly used. This practically precludes working in the hard limestone, so that the old workings show the approximate shape of the ore-bodies formerly met with. They are of necessity extremely irregular, varying from what are no better than rabbit-burrows, along which it is difficult to crawl lying flat, to large irregular chambers, which fortunately keep open without timber or stowing.

When any exceptionally large mass of limestone is met with in the breccia, the ore all round has to be laboriously dug out, and the boulder left underground, but with this exception, everything broken is taken outside for treatment—no very difficult matter where each man breaks only a few pounds of stuff per day. Recently, blasting with gunpowder has been introduced in a few cases, to break particularly troublesome masses of waste rock; the holes being bored by ordinary jumpers. But the Hazara miners are not paid for deadwork and dislike the extra expense; and the Afghan officials perceive that the price paid for the ore could hardly be lessened, even if gunpowder were provided free of charge, so it is not much used at present.

The miners' lamps are of baked clay or dug-out steatite, of the old open Roman pattern, and are carried in the hand or rested on projecting corners of the rock: cottonseed or mustard oil being burned, with a wick of cotton rag. The writer, with his assistant, Mr. Henry Middleton, and the native officials were provided with torches, formed by wrapping oiled rag around a short stick. No attention is paid to the ventilation, so that in places where narrow and tortuous leaders (a foot or so in width) are being worked the heat is great and the air very foul. But the miners do not seem to suffer in health, as they arrange so that no man works for long in the worst places. Moreover, the 20 days of freedom between each 10 days of work gives time to recoup, as well as to cultivate the land around their villages, on which their living depends. In one part of the ancient workings an ingenious contrivance for improving the

ventilation was noticed: the incline from the surface being made double, with a narrow bar of ground between, pierced at intervals as required.

No account is kept of the amount of stuff broken underground; but the amount of washed lead-ore bought daily by the smelting-works is about 30 cwts.; and, estimating that this represents four times its weight of stuff brought out of the mine, it appears that each man only breaks 20 to 25 lbs. of stuff per day—an amount surprisingly small, even when the toughness of the ground, the inferior tools, and the extreme narrowness of many of the workings are considered. It must, however, be remembered that each man has to crush and wash, as well as to break, his own ore.

Little or no handpicking is possible, so the ore brought out of the mine, which is already mostly in small pieces, is crushed by hand: a large stone being used as a pestle, on a flat beaten bed or another flat stone. It is then panned in a smooth wooden dish of about the same shape as an ordinary gold pan, but somewhat smaller (about 1 foot in diameter); the lighter waste, together with most of the slime galena, being washed away and lost. There is a certain amount of middle product, which is crushed and washed over again; but most of the concentrate is galena, fairly free from limestone and clay, but contaminated with a considerable amount of iron pyrites. It assays about 68 per cent. of lead and $1\frac{1}{2}$ ozs. of silver per ton.

Small amounts of lead-ore are also obtained from the Chilan mines among the rough hills a few miles to the south of the Ghorband valley. The rocks here are clay-slates and siliceous grits, lying nearly horizontal: these being cut through by many narrow veins, striking north-west and dipping 70 to 80 degs. south-west, corresponding to one of the jointings of the grits. These veins are generally only a few inches wide, the vein-filling being solid, large-bladed galena, with angular fragments of the country rock. The richer parts of the veins are followed up or down, wherever they may lead; and the workings are, in consequence, so narrow and tortuous that it is often most difficult to crawl or squeeze along them. The only tools used are hammer and gad, so that a pinch in the hard rock is an effectual bar to further progress; and for the same reason cross-cutting underground is unknown, though some of the veins are only 2 or 3 fathoms apart. The writer saw work being done on a vein less than 2 inches in width: some of the slaty wall-rock being perforce taken with the vein-stuff; but the average would perhaps be 6 to 8 inches. In a few places, especially where the rock is siliceous grit, and the workings have an upward trend, the old process of fire-setting is used: fires being lit against the face so that the flames play upon the rock, and cause it to split off in

flakes. The mines being at an elevation of nearly 10,000 feet, there are no trees, and the only fuel obtainable is dried camels' dung, and a small prickly bush, which is common on the stony mountain-slope, and which burns so quickly that it has to be continually replenished. The main objections to the process are the heat and smoke, all products of combustion having to find an outlet by the same tortuous channel, often less than 3 square feet in cross-section, by which the men enter. The ore is brought to the surface in skin bags, which are pushed or pulled along by the miner.

As already stated, the washed ore is bought from the miners at the rate of 3d. per 16 lbs., or about 35s. per ton; and it is smelted under the direct supervision of the Afghan officials, by men who give their whole time to the work, and are paid at the comparatively high rate of about 4d. per day. It is done in very small furnaces and at widely scattered centres, on account of the difficulties of procuring fuel: the six furnaces at the mine treat only a quarter of the total output, and are considered mainly as a means of controlling the quality of the washed ore, assaying being unknown. The furnaces used resemble the Scotch hearth in principle, and are very simple in construction. A rubble wall, 5 feet long, 5 feet high, and 18 inches thick, is built along the hillside, in a position best adapted for carrying off the fumes by the wind, and a rough roof is carried from the top of the wall back to the hillside to shelter the workmen producing the blast (see Fig. 2, Plate XIII.). The actual furnace is a rounded cavity in the bottom of the wall, lined with refractory steatite-clay, 18 inches by 14 inches by 7 inches, of the form shown in the sketches (see Figs. 3, 4, and 5, Plate XIII.), and it is worked entirely from the front. There are two tuyères of clay, $\frac{3}{4}$ inch in diameter; the blowing apparatus being two sheeps' skins, each fixed to a tuyère by the neck and split open at the other end. The split ends of each skin are turned around two sticks of wood, by which they can be grasped, and opened or closed with one hand. A workman sits between the skins, taking one in each hand, and forcing them alternately backwards and forwards, the split ends being opened on the back stroke and closed when the extended skin is being forced forwards again, thus producing an intermittent or pulsating blast. The work is so hard that two men are necessary to keep a furnace going, relieving one another at short intervals. It is characteristic of the Afghan want of ingenuity that although there is ample water-power in the valley close at hand, this cumbrous and inefficient method has never been improved upon.

The charge of a furnace is 20 lbs. of ore, which is thrown damp into

the hearth, still hot from the last charge, and a few sticks of wood are thrown in on top. The fuel used is exclusively dry soft wood, such as poplar and willow; harder woods like oak, mulberry, and walnut, being said not to answer. It is split into pieces, 10 inches or 1 foot long, and 1 or 2 inches thick. When these have kindled, a gentle blast is started, more wood being constantly added, and full blast quickly got up. The ore is frequently stirred and worked from the front with a thin poplar branch, and it is always covered with more or less wood. The flames and smoke come out in the front, much lead being lost in the fumes, which coat the front wall with yellow and orange incrustations and the ground around with fine, grey smoke; and, as is to be expected, the workmen suffer a good deal from lead colic. In $1\frac{1}{2}$ to 2 hours the smelting of a charge is finished, the lead being reduced and the molten bath covered with a dry sinter in small lumps, containing many metallic shots. All is now scraped down with the stick, and the slag repeatedly and thoroughly stirred into the molten lead. Then the blast is stopped, the slag is scraped out, and the lead is ladled with a shallow iron ladle into little rounded depressions in the ground; the hearth being at once cleaned, when it is ready for a new charge of damp ore.

Each furnace smelts six such charges, or 120 lbs. of ore, per day of 12 hours, and it is worked by four men—one smelter, two blowers, and a man who assists generally by splitting wood, etc. Each charge (20 lbs.) of ore requires 24 lbs. of dry wood, and yields $7\frac{1}{2}$ lbs. of good soft lead, or $37\frac{1}{2}$ per cent. out of 68 per cent. present. The balance is lost in fumes and in the slags, for although the latter are crushed and washed for metallic shots, they still contain as thrown away nearly 50 per cent. of combined lead. The ore from the Chilan mines is nearly pure galena, with quite 80 per cent. of lead; but when crushed and smelted in the same way it yields only half its weight, or 50 per cent. of metal. This ore contains from 15 to 20 ounces of silver to the ton, but so little is mined that it is disregarded—indeed the separation of silver from lead does not seem to be known to the Afghans.

The total cost of a ton of smelted lead from these mines is therefore as follows :—

	£	s.	d.
Mining, crushing and washing $2\frac{2}{3}$ tons of clean ore at 35s. per ton	4	13	4
Smelting—Labour, 1s. 3d. per 45 lbs. of lead ...	3	2	3
Fuel, 144 lbs. wood per 45 lbs. of lead ...	2	4	3
Superintendence, etc.	0	10	2
Total cost per ton of lead	£10	10	0

The total output being about 14 tons of smelted lead per month.

There are probably few places in the world where so many men are at work with so trifling a result, and at first sight no better field for the introduction of improvements could be found. But the introduction of European methods of mining would be impossible for political reasons. At present the Hazara miners consider the mines to be their own property, where they can work how they please, being bound only to extract a certain amount of ore at a fixed price, and any interference with the system would be resented. Moreover, European methods would mean the creation of a body of skilled men in constant employment, who would have to be paid a living wage of say 4d. per day, and so the cleaned ore could probably never be obtained for less than the present very low figure. It is true that the output could be vastly increased, but these and other mines already produce enough for the needs of the country, almost confined to the making of bullets; and there is a disinclination to export metal. Probably the most practical improvements in Afghan mining would be the introduction of the Cornish poll-pick, and the extended use of blasting for breaking the harder and narrower parts of the lode; while simple machinery could easily be made locally for crushing and dressing the ore, so as to avoid most of the present loss in slimes.

But it is in the smelting that the greatest economies could be effected. By increasing the size of the hearths and using a water-power blast the present absurd expenses of labour and fuel could be lessened; and with flues to condense the fume, and a slag-hearth to re-smelt the slags, a fairer percentage of metal might be saved. Brushwood is fairly cheap and abundant, so an adaptation of the smelting-furnaces used at Linares in Spain might be even more successful.

It will be understood that the unsettled political condition of the country has as much to do with the backward condition of mining as the ignorance of the people. There is evidence that things were not always so bad as they are now; thus, small amounts of fused slags, poor in lead, found at the surface, show that smelting was at one time better understood. But no ruler latterly has been sure enough of his position to give much attention to mines, and a class of professional miners would fare badly in time of war. It is only as the country becomes more settled politically that the mining industry will be likely to improve.

The following paper by Mr. A. G. Charleton on "The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment, Part VI." was taken as read :—

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part VI.—Gold-Milling.—*Continued.*

Practice in Dakota.†

The gold of the district is found in quartz and pyrites finely distributed through vast masses of mica and amphibole-schists, argillites, and phyllites, and also impregnating the schists themselves. The gold-belt embraces the sections of Lead City, Terraville, and Central City. The principal associated mineral is iron pyrites, with some arsenical pyrites, garnet, and asbestos.

The ores from open cuts and the upper levels are more free-milling than those from the mine workings below the water-line. Hence the mills running on oxidized ore have tailings valued as low as 1s. 0½d., while tailings from unaltered ore run up to 9s. 4½d. per ton. By watching the pulp when it flows down the plates, it can readily be determined whether it comes from the higher or lower levels. In the former case it will generally have a brownish-red colour, and in the latter it is of a bluish-grey. The amount of free gold in the ore varies with depth, and probably 16s. 8d. per ton is a near average of its value.

To determine the amount of free gold in the ore the following method is practised :—Samples are taken daily from the various workings in the mine and sent to the sampler, who crushes and pans them and estimates the gold in each pan. Every valuation thus made is booked, and at the end of the month the average is taken, and compared with the output of the mill, and the amount of gold recovered thus approximately determined. The mode of operation practised by the sampler is very simple. The sample, weighing say 10 pounds, is emptied into a 4 gallons bell-shaped mortar (13½ by 12½ inches). From it 2 pounds are then

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351 ; vol. v., page 271 ; vol. vi., pages 69 and 295.

† The writer is largely indebted in this section to a paper by Mr. H. O. Hofman, on "Gold-Milling in the Black Hills."—*Trans. American Inst. Min. Eng.*, vol. xvii., page 498.

transferred into a second mortar of the same size with a wooden lid, and pulverized wet to a fine pulp by means of a small steam stamp, which is in reality an old power-drill fitted up for the purpose. When sufficiently fine (as judged by the ring of the pounding-stamp) the pulp is panned until all the pyrites and heavy sands are washed off with the tailings, and only the free-gold remains behind. The sampler of the Homestake Company pans from 50 to 55 samples per day. Great skill is acquired in thus estimating the value of the ore, the sampler being able to make eight or ten valuations per hour. As these are the only determinations made, the amount of non-free-milling gold which enters the mill is not known.

The percentage of sulphides from several determinations that have been made varies from $2\frac{1}{2}$ and 3 to 6 and even 10 per cent. The assay value of pure concentrates freed from rusty gold, or gold included still in the quartz, has been shown to vary from 16s. 8d. to £16 17s. 6d., the average being about £5 4s. 2d. per ton.

As the ore is finely disseminated throughout the entire vein-matter, comparatively little sorting can be done in the mine. There occur, however, in many parts of the veins igneous intrusions, locally called porphyry, which form barren horses. When the Nevada system of timbering in square-sets was exclusively in use no distinction was made between mill-rock and waste, but it was considered cheapest to run it all through the mill. Latterly, however, it has become the custom to fill the chambers formed by this timbering with waste, and to hoist the excess to the dump; but large quantities of it are still crushed.

The auriferous lodes and gravels were discovered in this district in 1876, when a rush to the Black Hills took place.

The seven principal mills are the Homestake, Golden Star, Highland, Deadwood, Golden Terra, Father de Smet, and Caledonia. The first six are owned by five separate companies, but are all under the management of the Homestake mill superintendent, and all the working details are much on the same model. The Caledonia mill works on different rock, and differs from the others in details of plant.

The crushing is all done by rock-breakers and stamps. The ore arriving at the ore-floor at the highest level of the mill is discharged from side or bottom-dump cars over grizzlies, which divide it into lumps (going to the breaker) and fines (falling into the bins). The lumps are crushed in the rock-breakers and join the fines, and the mixed product, passing through shoots, goes to the automatic feeders, which deliver it to the stamps.

The Caledonia mill has blankets on the lower end of the apron-plates, below the copper plates, to catch any coarse heavy particles. In the other mills the pulp passes direct from the apron-plates to the mercury-traps, and through them on to sluice-plates. From the traps placed at the end of these the pulp runs into one main sluice, which may have again one or more mercury-traps before the pulp is allowed to finally run to waste.

Amalgamation begins in the mortar (which is provided with inside copper plates, mercury being added to the box at intervals), and is continued outside on the apron-plates; any mercury or amalgam escaping these appliances being caught in the mercury-traps* lower down, which are used to supplement them.

The aim in Dakota is to crush rapidly to the desired degree of fineness and arrange the amalgamation so that it shall be adapted to the large amount of pulp produced. The distribution of power in the mills is of three different types, represented respectively by the Homestake, Golden Star, and Highland mills.

In the Homestake mill the continuation of the engine-shaft forms the one-line shaft of the mill, and is placed on the battery-sills. This is a cheap construction, and gives a solid foundation for the boxes in which the line-shaft rests. The shaft is kept in line by the even pull of the long belts on each side of it, running at an angle of about 30 degs. between the bins in the centre of the mill. The disadvantage of having to stop the mill if anything happens to the line-shaft may be dismissed as insignificant. For minor repairs, however, it is in an obscure place, running behind the batteries.

In the Golden Star mill the power is transferred from a small main shaft to two line-shafts on the cam-floor, which is nearly on the same level. This arrangement is usual in Pacific coast batteries.

In the Highland mill the small main shaft is placed between the cam-floor and the battery-floor, and is connected with two line-shafts† placed on the battery-sills behind the mortars, which are set back to back with the bins between them. Two lines of shaft are used, in consequence of the power required for a 120 stamp. The disadvantage is that the pull of the belt on one side only of the shaft, has a tendency to drag it out of line and cause excessive wear and tear.

* An illustration of this apparatus will be found in the *Eighth Annual Report of the California State Mineralogist*, page 711.

† These are coupled up in sections, 7 inches in diameter near the engine, reduced at the opposite end to 4 inches.

As to placing the line-shafts in front of the batteries on the cam-floor (as in the Golden Star mill), or behind them on the battery-floor (as in the Highland), there is a diversity of opinion. The former arrangement gives the best light, and makes the shafting easily accessible. The power from the engine-shaft is transmitted to the line-shafts, and from these to the cam-shafts by horizontal belts which require no tighteners and last longer. On the other hand, the boxes of the line-shafts rest on traverse sills on the cam-floor which, although braced and otherwise strengthened, do not afford as solid a foundation as when fixed on the battery-sills. On the whole, however, this disadvantage appears more than counterbalanced by their greater accessibility and the smaller wear and tear of belting.

On the battery-sills behind the mortar, the line-shaft is in darkness and exposed to the trickling of water and the abrasion and dirt of fine ore falling from the feeder-floor, whilst the belts connecting the main and line-shafts and the latter with the cam-shafts, are short and highly inclined, requiring powerful tighteners.

The relation of the horse-power of the engine to each stamp averages about 1·7 to 1 for the seven mills. This low figure is due to the large number of stamps in each mill (80 to 120) which is much above the ordinary number.

The Father de Smet mill is built with the batteries on opposite sides of the building face to face, an arrangement which is said to have the disadvantage of rendering it rather dark.

The Golden Star and Highland mills have each 120 stamps, and the Father de Smet mill 100 stamps of 850 lbs., dropping 85 times per minute, with a fall of 9 inches, arranged in batteries of five. The Homestake, Deadwood, and Golden Terra mills have each 80 stamps, with the same weight of stamps and height and number of drops. All these mills crush on the average about the same weight of ore per stamp head, viz., 4·5 tons per 24 hours. The Caledonia is a 60 stamp mill with 850 lbs. stamps, dropping 12 inches, 74 drops per minute, and it only puts through 3·3 tons of ore per 24 hours.

The water supply is furnished by ditch companies at 50 to 57 cents per stamp per diem. A regular supply is an essential for milling. When it becomes scarce in winter it is supplemented by pumping from the Homestake and Deadwood Terra shafts.

The Highland mill would be obliged to stop four months each winter if the tailings of the Homestake, Golden Star, and Highland mills were not settled and the clear water pumped back into the mill supply-tanks. The

method by which this is done is of interest. The tailings of the three mills named are discharged into Gold Run creek. A little way down, where the creek broadens it is closed by two dams, one below the other, forming an upper and lower reservoir. The former overflows into the latter, which is four times its size, and this in turn overflows into the bed of the creek. The dams consist of cribbing filled with waste rock, lined on their upper sides with planking to make them watertight. Down the middle of the face of the upper dam runs a wooden box or launder, three sides solidly planked, with the fourth open ; when the reservoir is to be filled the box is closed by heavy transverse strips of planking. The object of a number of pieces is to discharge the water gradually, which is done by removing the pieces one after the other as the water is lowered, so that the sands may be kept in suspension and carried through the culvert. Were the box open from the bottom or to its full height at once, the sands would be carried into the culvert in such quantity as to close it. This culvert in which the box ends, passes through the dam and under the large reservoir and lower dam to the bed of the creek below. The lower dam is arranged in the same way. When the reservoirs are not in use the water of the creek passes off through the culvert. When they are to be filled the boxes are closed, and the water accumulates in the upper reservoir until, after 6 hours, it overflows, leaving all the coarse sands in the upper reservoir, and carrying only the finer slimes, which settle in the lower one. From this the clarified water is pumped at the rate of 60 cubic feet per minute into the Highland tank 200 feet higher.

The coarser sands are removed from the upper reservoir every 24 hours. In order to do this the traverse planks closing the discharge are removed one after another, and the water passes off, carrying the sands with it. As this process takes 4 hours, and the filling 6, there are 14 hours of overflow into the lower basin, where the slimes settle. These are removed once in two months in the same way.

In Queensland, Australia, where water has to be rigorously economized, a reservoir is frequently built in a suitable position at the side of a creek, so as to enclose a large area. Openings are then made in the artificial embankment which takes the place of one bank of the creek, and lock-gates are thrown across the stream forming an extension of the embankment at the lower end of the reservoir. In the rainy season these gates are closed temporarily so as to raise the water to the level of the openings in the embankment, through which it finds its way into the reservoir until the latter is filled. The water accumulated in this way is used as a

reserve in the dry seasons, at which time, if the sides of the creek are abrupt enough to admit of doing so, large dams of sand formed of tailings may be thrown across the bed of the stream to back up any water that can be accumulated behind them. The accumulated tailings are washed away in the rainy season when the flood-gates are opened. The water, notwithstanding that it carries considerable silt, has often to be used over and over again by pumping it from the reservoirs with a tailings-pump, which, in order to prevent the plunger from being cut to pieces, has a small stream of water introduced under pressure by a small pipe into the plunger-case.

In some Queensland mills, the system adopted is to construct a breast-dam running right across the creek, with a gate in the middle through which the silt which accumulates, partly in times of flood, partly from settling the tailings, can be flushed out. At the end of the rainy season, it is filled with water, and the mill draws its supply from this reservoir by means of a pump. The mill-tailings are discharged immediately below the reservoir through box-launders put together in short sections, the lower end of one entering the top of the section below. After a heap of tailings has accumulated against the back of the dam, at the side of the creek over an area of 5 or 6 square yards, an accumulation which can be facilitated by changing the position of discharge of the launders, the flow is diverted to one side until the heap has drained and settled enough to allow of a shallow pit about 1 foot deep being dug in the surface of the heap (which is roughly levelled), leaving a wall of sand (tailings) round the edge. The mill tailings are then allowed to run into this temporary basin, and the sands settle in it, whilst the partially clarified water is returned by a short launder through a hole in the face of the dam (near the top) into the reservoir. Whilst the first basin is being filled up, the tailings-man is engaged in digging out a second one at the side of the first, into which the tailings are diverted when the first one becomes partly filled with sand. The heap forming the foundation of the basins is constantly extended outwards by throwing with a shovel the sand from the basin to the outside of the wall of sand, which forms a natural talus sloping outwards. It is of the greatest importance when this method is used to keep the level of the basins as high as possible, with a slight rise outwards as they are extended into the creek, otherwise the water cannot circulate backwards at a sufficiently high level to enter the reservoir at the point where the opening is made for this purpose.

The fuel for the mills of the Homestake management is supplied by the Black Hills and Forte Pierre Railroad Company. This road, with

about 30 miles of 3 feet gauge track, runs along the divide between Gold Run and City Creek, terminating at a point about 14 miles south of Lead City. The whole section was originally heavily wooded, but has been quite denuded by the constant demands made upon it. The railroad is very winding, and is quite a feat of engineering. It runs down the slope into Whitewood Creek and up the opposite height, till it finally reaches the point where timber is still to be obtained. This road, as soon as it is open in the spring, is employed in transporting the timber which has been cut and stored along its line, and is in constant use till it becomes blocked with snow, generally from January till April. It has three branches towards the three towns where the mills are situated, and communicates directly with these by long wooden shoots, down which the timber is discharged. These shoots are 700 to 1,500 feet long, running down the slope of the mountain, they are 25 inches broad by 12 inches deep, and are made of 4 inches planking. The bottom and 9 inches of the sides are lined with $\frac{1}{4}$ inch iron plates. The fall of the shoot is 6 inches to the foot until the curve begins, when it is $4\frac{1}{2}$ inches. This continues to the nozzle, which is elliptical. When the shoot is in use a small current of water is passed through it, to prevent the iron from becoming too hot and to act as a lubricant. The cord wood unloaded at the top of the shoot passes down the incline with great velocity. At the nozzle it is deflected from its course, and through the momentum obtained in the downward passage it shoots up into the air and drops some distance off on to the wood-pile. In order to discharge the wood on a large area and to stack it, the nozzle is made movable.

The price of wood in the district is £1 5s. per cord. The water is warmed, as elsewhere mentioned, to keep it from freezing in winter, and fireplugs with hose-attachments are placed at intervals in the mills as a safeguard against fire.

To reduce the cost of repairs, which is a heavy item, the Homestake Mill Company has a foundry where rock-breaker shoes and dies, pitmen and toggle-plates, mortars and dies, boss-heads, tappets, thimbles for fingers, cams and hubs of cam-shaft pulleys and shaft-boxes, etc., are cast from Nos. 1 and 3 foundry-iron and worn-out castings. The castings are made in sand, with the exception of the rock-breaker shoes and dies and faces of battery-dies, which are chilled. All the necessary repairs are executed in the machine-shop, which is a very complete establishment. The six Homestake mills use the same patterns for all parts requiring frequent renewal, which reduces the amount of material kept on hand, and labour and cost of repairs.

The Grizzlies relieve the rock-breakers of the ore which does not need crushing. They are 3 to 4½ feet wide, 10 to 14 feet long, and set at an angle of about 40 degs., or a rise of ⅞ inch per foot. They are made generally of wrought-iron* bars 1 inch wide and 2 to 4 inches deep, held 1½ to 2 inches apart by three or four, sometimes five, 1 inch iron rods, provided with thimbles at proper intervals to keep them apart. The Father de Smet grizzlies, which are 4½ by 12 feet, with 24 bars 1 inch by 2 inches set 1½ inches apart, weigh 2,040 lbs. The grates last about four years.

Rock-breakers.—No. 5 Blake rock-breakers are mostly in use, with jaw (opening 9 inches by 15 inches) set to crush from 1½ to 1¾ inches, run for 20 hours out of the 24 ; this size is calculated to serve 20 stamps. If one-fourth of the ore passes through the grizzlies and 20 stamps crush 90 tons in 24 hours, the amount crushed by one such Blake breaker in 20 hours is 67½ tons or 3·4 tons per hour. This small figure as compared with the nominal capacity of the crusher, *i.e.*, 7 tons, is due to the delay in breaking up the ore so as to enter the jaws. The advantages of the Gates crusher in this respect have been previously mentioned. The No. 6 Gates crusher in use at the Caledonian mill is said by Mr. T. L. Skinner, the superintendent, to have saved him £5 12s. 6d. per day.

Mr. Hofman advocates for a large mill a still larger Gates crusher (No. 8 with receiving openings 18 inches by 48 inches), set to crush coarse, discharging into two No. 6 crushers set to crush fine. Thus the largest pieces of rock that any man could handle would pass direct into the crusher without hand-breaking.

Ore-bins.—The ore-bins are triangular in section, with one vertical side facing the battery and reaching down to the cam-floor. Just above the latter are the openings (one for each feeder) through which the ore passes down into the shoots terminating in the hoppers of the feeders. The discharge can be regulated by a sliding-gate, with a rack worked by a pinion keyed to the shaft of a small hand-wheel. In a double mill the inclined bottoms of the two bins diverge, leaving an open space between them having the shape of an inverted V. The bottoms of the bins, 3 inches thick, are made of 1 inch board, running lengthways, with 2 inches planking set crossways on them, but this is not the best arrangement. The bottom and sides are strongly braced with timber. The upper part of the bottom on which the ore drops from the grizzlies and crushers is

* Hard cast-iron is being introduced for this purpose in some mills.

lined with iron to prevent it from wearing out faster than the lower parts, which last five to six years. There are no compartments or divisions to direct the ore to the discharge-openings, as they are unnecessary. Ore-bins should be given as large a capacity as possible, so that, in case of accident at the mine or to the rock-breaker the mill need not be stopped. The Highland mill has the largest bin capacity of the group under review, except the Father de Smet mill. In the Highland mill the horizontal distance between two sets of batteries is 46 feet, and the vertical height is 22¾ feet. In the Father de Smet the apron-plates are overshadowed by the bottom of the bins, which neutralizes the advantage of facility of supervision claimed for this arrangement, by rendering the building dark.

Feeders.—The feeder may be arranged so that the lip of the feeder reaches into the feed-slit of the mortar, but at the Caledonia mill they discharge with a small inclined iron-lined apron which leads to the mortar. By this arrangement a little more room is left between the feeder and mortar, and the feed-opening can be longer and narrower, distributing the ore more uniformly under the stamps. The Homestake mill mortar feed-opening is 24 inches long and 4½ inches wide, while at the Caledonia it is 52 inches long and only 3 inches broad, occupying the whole length of the mortar. The Hendy Challenge and Tullock automatic feeders are in use, the Challenge feeder being the most desirable for wet ores.

Both right and left-hand feeders are used, the bumper-rod standing between stamps 1 and 2 or 4 and 5. Some of newer pattern are made with the bumper-rod next the central stamp (central feeders). The rod is guided from the cam-floor, passing through a hole in a board fixed to it. The sheet-iron plate below the Tullock feed wears out quickly (with Homestake ore in two years), but is cheap and can be patched or renewed by a blacksmith. The circular cast-iron carrier-table of the Challenge feeder lasts seven years with the same class of ore, but is costly; and if anything goes wrong with the gearing, it requires a fitting shop and machinist for its repair.

Foundations, etc.—A good foundation is the starting-point of every piece of good engineering work, and nowhere is it more necessary than in a stamp-battery. A rectangular pit, 11 to 14 feet deep, is excavated to receive the mortar-blocks, made sufficiently long and wide (4 feet by 6½ feet) to leave a space of about 24 inches round the block. The bottom is then carefully levelled and some sand tamped down, or concrete is run

in. On this are placed two layers of 2 inches planks, spiked cross-wise to each other, and then the planks which form the mortar-block. The latter used to be placed in the pit, and the uneven tops were afterwards sawn off level. Now care is taken that this 4 inches wooden floor shall be accurately horizontal, and that the distance between it and the bottom of the mortar shall correspond with the length of the mortar-blocks. The top of the block is planed. By employing this flooring the time required to replace a mortar-block is reduced from six or seven days to five.

The mortar-blocks used in Dakota consist of planks from 11 to 14 feet long, depending on the depth of pit, of varying width, and not more than 2 or 3 inches thick, as it is difficult to find wood of greater thickness and yet sound throughout. They are spiked together, and fastened above and below with binders bolted to each other by transverse rods, the upper binders (8 by 12 inches) being even with the top of the mortar-block, and the lower binders (12 by 12 inches) 3 feet lower down. The space round the block is then carefully filled with rock and tailings up to the level of the mud-sills, which are about 4 feet below the bottom of the mortar. When the top of the block has been planed off level, a sheet of rubber-cloth $\frac{1}{4}$ inch thick is placed over it and the box put in place. Through the four holes in the flanges on each side pass 8 bolts from 3 to $4\frac{1}{2}$ feet long, and $1\frac{3}{8}$ to $1\frac{1}{2}$ inches in diameter, by which the mortar is held down.

In placing the planks forming the block and adjusting the holding-down bolts an improvement has been introduced at the Homestake mills. The planks which always stand on end were formerly so placed that their width was parallel to the short side of the mortar. The holes for the 8 bolts were then bored into the block from above, and at a suitable distance below recesses were chipped out to receive the nuts which held the lower ends of the bolts. Now the planks are so placed that their width is parallel to the long sides of the mortar. The bolts are only threaded at their upper ends, and end in an eye at the bottom, 2 inches bolts pass horizontally through these loops from side to side of the block, the planks on the two sides of the block where the bolts pass down, being cut out to receive them.

In addition to the mortar being more securely and evenly tied to the block, this makes the replacement of a mortar-block if required much easier. The pit need only be dug out in front of the mortar, and when the front binders have been removed it is easy to tear out the planks one after another with pick and adze. In putting in the new block, the two out-

side rows of planks, with places cut to receive the bolts, are kept ready so that only four horizontal $2\frac{1}{2}$ inches holes for the rods need be bored when the planks have been spiked together.

Battery-posts are usually made of 12 by 24 inches timber with recesses cut for the boxes of the cam-shaft. They are set on the short sides of the mortar and are independent of the mortar-block, standing on the cross-sills which rest at right angles across the mud-sills; they are tied together by the upper and lower guide-timbers and at the foot by two beams bolted to them, running parallel with the long side of the mortar-block, and let into the cross-sills.

The frames are braced either from back or front of the battery, depending on the way in which power is transmitted to the cam-shaft. When the line-shafting is at the back they are braced by inclined struts at the back between the posts and cross-sills. When the cam-shafts, however, receive their motion from line-shafting in the cam-floor in front, the posts are braced by horizontal cross-beams, resting on a line of posts in front, and projecting beyond them, which are strutted in the angles, and tied together by a longitudinal cap-piece.

Guides.—The stems work in two sets of guides fixed to the guide-timbers which tie the battery-frames. The upper guides are above the tappets, the lower ones below the cam-shaft. At the Homestake mill the centre of the lower guides is $17\frac{1}{2}$ inches above the top of the mortar; the cam-shaft is 4 feet higher from centre to centre, and the centre of the top guides is 3 feet $10\frac{1}{2}$ inches above this. At the Caledonia mill the distance from the top of the mortar to the centre of the lower guides is $16\frac{1}{2}$ inches, from these to the centre of the cam-shaft is 3 feet $4\frac{1}{2}$ inches and between the cam-shaft and the top of the guides, from centre to centre is 4 feet $2\frac{1}{2}$ inches. Each set of guides is provided with liners (guide-blocks) consisting of two strips of 4 inches pinewood, 16 inches deep, provided with semicircular grooves for the stems. When new, small wooden strips are inserted between the back and front of the blocks to hold them slightly apart, these strips are removed as the grooves become worn, bringing the faces of the guide-blocks nearer together. When still further worn the faces are planed to diminish the depth of the grooves, so that the stems may not be too loosely held. Each set of guide-blocks is secured to the guide-timbers with eight $\frac{7}{8}$ inch bolts. The grooves and the guide-blocks are lubricated with a preparation of black lead and linseed oil, mixed warm so as to form a soft paste. Oak guide-blocks last 18 months, pinewood only four.

Mortars.—Two kinds of single-discharge mortars are used in the district, each being a solid casting. The sides and bottom are so thick that a lining is dispensed with, but as the feed-opening is approached the thickness rapidly diminishes. The discharge-side projects somewhat, and the other sides are vertical. The top is closed by two pieces of 2 inches planking which rest on lugs $\frac{3}{4}$ inch wide, cast on the inside of the box 2 inches below the top. These planks have, as usual, five semicircular recesses cut in them, which, when placed together, form holes for the stamp-stems to pass through. Two smaller holes are also bored in them, for two 1 inch water-supply pipes placed between stamps 1 and 2, and 4 and 5.

The mortars are set close together in pairs, 10 stamps being operated by one cam-shaft, and a passage-way is left between every two pairs of batteries.

The water-supply is furnished by a 3 inches main-pipe in front of the batteries, from which a 2 inches stand-pipe passes upward between each pair of batteries. A 2 inches horizontal pipe is connected with the stand-pipe from which four 1 inch pipes branch off at right angles, two for each mortar. A 1 inch pipe at each passage-way close to the mortar, connected with the 3 inches main-pipe, serves for the attachment of a hose for cleaning the apron-plates and other purposes.

The points of difference between the Homestake and Caledonia mortars lie in the dimensions of the lower part of the box, and the number of the inside amalgamating-plates. The Homestake mill mortar weighs 5,400 lbs., is $54\frac{1}{2}$ inches high, and $54\frac{3}{4}$ inches long. The feed-opening ($6\frac{1}{2}$ inches below the top) is 24 inches long, $4\frac{1}{2}$ inches wide, and 7 inches deep, and continues the same length inside; the incline over which the ore falls being extended to form a lip $4\frac{3}{4}$ inches wide and $1\frac{1}{4}$ inches thick, projects into the mortar so as to discharge the ore against the upper half of the stamp-head. The lower edge of this lip is 14 inches above the bottom of the mortar. As it wears out quickly, it is cast thicker in the Caledonia mill mortar.

The discharge-opening in front of the Homestake mortar is $15\frac{1}{2}$ inches from the bottom, $48\frac{1}{2}$ inches long, and $21\frac{3}{4}$ inches high. The frame-seat is inclined outwards about 10 degrees from the vertical, and there are grooves at its ends to receive the chuck-block, screen-frame, and curtain, which are held in place of keys. The chuck-block is secured also at the bottom by two horizontal keys supported by lugs cast on the lip of the mortar. The bottom flanges are 3 inches high, and 5 inches broad; the bottom is $7\frac{1}{2}$ inches deep, and the sides at the foot of the dies are $3\frac{1}{2}$

inches thick. The width inside at the bottom is $10\frac{1}{2}$ inches,* its length is 50 inches, and height to issue of mortar, *i.e.*, the bottom of the discharge-opening, $8\frac{3}{4}$ inches. The inside width of the mortar at this point is $13\frac{1}{2}$ inches, and at the top of the discharge-opening it is 20 inches. At the top of the mortar it is 16 inches, and the total inside height is 47 inches. The casting is $\frac{3}{4}$ inch thick from the top down to the feed-opening on the sides and front, but the back is a little thicker. The life of a mortar is about four years.

The Caledonia mortar weighs 5,700 lbs., is $57\frac{1}{2}$ inches high, and 54 inches long. The feed-opening begins $15\frac{1}{2}$ inches from the top, is 3 inches wide, 11 inches deep, and extends the whole length of the mortar, with a strengthening rib in the middle. Where it enters the mortar it is $50\frac{1}{2}$ inches long, and $7\frac{1}{2}$ inches deep, with a lip $2\frac{1}{2}$ inches thick, and 8 inches wide, measured on the incline. This discharges the ore towards the head of the stamp, and protects the amalgamated plate below. The front discharge-opening (50 inches by 17 inches) is 20 inches above the bottom of the flange, and is inclined forward about 10 degs. The grooves on the sides receiving only the screen-frame and curtain are of simpler construction than the Homestake mortar. The lugs for the horizontal keys are the same. The flange round the bottom is 3 inches thick and $4\frac{1}{2}$ inches wide. The mortar-bed is 7 inches thick and the sides at the foot of the dies are $4\frac{1}{2}$ inches thick. The width inside at the bottom is 10 inches, the length $50\frac{1}{2}$ inches, the height 14 inches to issue of mortar and pulp, where the width is 16 inches and increases to 19 inches at the top of the discharge. The top of the mortar is $13\frac{1}{2}$ inches wide, the total inside height is $50\frac{1}{2}$ inches, and the casting from the top to the feed-opening is $\frac{3}{4}$ inch thick. A mortar lasts six years, and wears out more at the ends than at the back. The feed-opening, for reasons previously recorded, is longer in the Caledonia than in the Homestake mortar, and its inside lip is thicker and wider in the former type, a difference which is necessitated by the use of the amalgamated plate below. For the same reason the Caledonia mortar is also made wider at the issue. The depth of the discharge-opening of the two types moreover differs. In the Caledonia mortar it is 14 inches, which represents the point of issue of the pulp, whilst in the Homestake mortar it is only $8\frac{3}{4}$ inches, as the issue is raised $16\frac{1}{4}$ inches above the dies by the insertion of a chuck-block, thus giving the shallower mortar the deeper issue.

Dies.—The dies are cast by the Homestake Company on the spot, using an iron between grey and mottled, the top of the cylindrical column

* Formerly it was 14 inches, but this reduced the crushing capacity.

being chilled. The footplate has bevelled corners, and is 10 inches long, $10\frac{1}{2}$ inches wide, and $1\frac{1}{4}$ inches thick. The column or boss is 9 inches in diameter, and 5 inches high. The level of the die is 10 inches below the discharge, which is over the chuck-block. The die weighs 121 lbs. (one-seventh the weight of the stamp), and lasts about six weeks, crushing 189 tons. By that time the boss has worn down to 2 inches from the footplate, and is slightly convex, its weight being reduced to 30 lbs., it shows a consumption of 48 lbs. of iron per 100 tons stamped.

The Caledonia mill purchases its dies, which are made of chilled white iron. The footplate is 10 inches wide by $9\frac{1}{2}$ inches long, and $1\frac{1}{2}$ inches thick. The boss is 8 inches in diameter and $5\frac{1}{2}$ inches high.

The dies in the Homestake mortar fill the bottom completely, those of the Caledonia only fit perfectly crossways, half an inch of space being left between them. From the bottom of the screen to the top of the die is 6 inches. The die weighs 160 lbs. (about one-fifth the weight of the stamp), and lasts three months, crushing 300 tons of hard rock. The boss is then worn to within 1 inch of the footplate. The worn-out die weighs 38 lbs., making the consumption of iron 40 lbs. for every 100 tons of rock.

Amalgamated copper-plates are placed along the entire length of the mortar. In the Homestake there is one plate only set in the discharge-opening. In the Caledonia a second one is used below the lip of the feed-opening.

In the Homestake mortar the chuck-block, consisting of a 2 inches planking bolted to the back of a $1\frac{3}{4}$ inches board, and extending from 2 to $2\frac{1}{2}$ inches above it, fills the bottom of the discharge-opening, the ends of the plank being held in the end-grooves outside. The inside upper edge of the block is rounded off, and over this and along the inside face a $\frac{3}{16}$ inch copper plate is fastened with iron screws. The recess in front of the chuck-block on the top of the front board ($1\frac{3}{4}$ inches wide and 2 to $2\frac{1}{2}$ inches deep) is taken up by the lower side of the screen-frame, between which and the front board a strip of blanket is laid to form a tight joint. The recess under the chuck-block back of the front board is filled by the side of the opening of the mortar.

The screen-frame is held in place by a vertical plate of iron bolted to the centre of the front board, with a horizontal wedge driven between them. The front board is faced, in the centre of the lower half and ends, with iron-plate to protect the wood against the two vertical and horizontal wedges which hold it to the mortar.

A strip of rubber cloth is tacked to the bottom of the chuck-block, in order to make a tight joint inside between it and the flange of the mortar-

opening. Two chuck-blocks of different heights are used, one 7 inches in height used when the dies are new, and one 5 inches high inserted when they are worn down 2 inches. The height of discharge is thus kept nearly uniform. Wooden chuck-blocks last six months.

Owing to the distance between the edge of the dies and the face of the chuck-block being rather small, viz., 2 inches, it was found in the Homestake mill that the sands driven violently by the water against the copper plate scoured off some of the amalgam. Mr. Graham, the millwright, has therefore replaced the 2 inches planking to which the copper plate is screwed, by a $\frac{1}{4}$ inch iron plate, to which the $\frac{3}{16}$ inch copper plate is riveted with copper rivets. The face of the $1\frac{3}{4}$ inches front board being covered with $\frac{1}{8}$ inch iron plate, the distance between the dies and the Graham modified chuck-block is $3\frac{5}{8}$ inches instead of 2 inches. This iron chuck-block lasts as long as the mortar, and more amalgam collects on it. Of the free gold recovered 55 per cent. is caught on this inside plate.

With wooden chuck-blocks the copper is removed when the block is worn out and reset on a new one, or they are scraped very carefully, put aside, melted, and sold.

The reason why the Caledonia mill has amalgamated plates at both the back and front is that the ore milled is not at all oxidized, making it more difficult to extract the gold. The aim, therefore, is to keep the pulp longer in the battery and present a larger surface of copper plate for amalgamation. The copper plate in front is 5 inches broad, and that at the back 8 inches. Both are made of $\frac{3}{16}$ inch copper plate bolted direct to the inside of the mortar. The lower edge of the plates is 9 inches above the bottom of the dies. Of the free gold recovered 60 per cent. is caught in the box.

Screens.—The Father de Smet mill uses No. 30 brass-wire screens, while all the other Homestake mills employ diagonal slot No. 7 Russian iron of No. $24\frac{1}{2}$ American wire gauge, weighing 0·987 lb. per square foot. The slots are $\frac{1}{2}$ inch long, and there are 8 to the inch. The punched surface is 48 inches by 7 inches, with a 1 inch margin, making the screen 50 inches by 9 inches. A screen lasts two weeks. The wooden frame is 4 feet 4 inches long by $11\frac{1}{2}$ inches deep outside, and has a strengthening piece down the centre. To fasten the screen in place, the lap is first tacked on to hold it in place, then a strip of rubber-cloth 2 inches wide is placed over it. Small holes are punched through the rubber and the lap of the screen, and both are nailed to the wooden frame, the burr facing the inside. The outside of the frame is protected by

three iron facings, $\frac{1}{2}$ inch by 9 inches by $\frac{3}{16}$ inch thick, fastened at the middle and ends with a couple of wood screws. Screens of aluminium bronze have been tried, and found so satisfactory that they are likely to replace Russian iron entirely.

The Caledonia mill uses No. 24 brass-wire screens, the thickness of the wire being No. 26, and the screen-surface, 48 inches by $5\frac{3}{8}$ inches. The screen lasts one week. It is fastened to a plain wooden frame, 53 inches by $12\frac{1}{2}$ inches, the horizontal sides being $3\frac{1}{2}$ inches wide, and the vertical sides $2\frac{1}{2}$ inches. Three wooden ribs, 1 inch wide, divide the surface into four panels, and keep it from bulging outwards. The attachments of the screen and frame are the same as at the Homestake mill, except that here it is keyed against the lower edge of the discharge-opening instead of resting on the board of the chuck-block. The Caledonia mill uses wire screens because, though its stamps drop 3 inches farther than the Homestake mill, the splash is not so great, owing to the greater width of the mortar.

The force of the splash in the narrow Homestake mortar is thrown entirely against the screen, while in the wider Caledonia type it is divided between the screen in front and the recess at the back, hence the slot-screen would clog. The upper part of the discharge of both classes of mortar above the screen-frames is closed either by a 1 inch board or a canvas curtain, or by a piece of old belting suspended from a lath. This hangs down and meets the screen inside the mortar. The curtain has the advantage over the board that the amalgamator can easily pass his hand inside, and remove chips of wood liable to choke the screen. In order to break the fall of the pulp (forcibly driven against the screen) on the apron-plate of the Homestake mill, a splashboard is fastened to the frame of the latter, to prevent any amalgam collected there being washed away. The Caledonia mortar has no splashboard, as the pulp does not pass the screen with sufficient force to endanger the amalgam at the head of the apron-plate outside.

Stamps.—The stamps weigh 850 lbs., and have about 16 lbs. to the square inch of crushing surface. The stem of wrought-iron tapers 6 inches at both ends, so that it can be reversed if broken. A stem lasts about three years at the Homestake mill, before new ends have to be welded on; it is 14 feet long, $3\frac{1}{8}$ inches in diameter, and weighs 340 lbs. The cast-iron head is 18 inches high, 9 inches in diameter at the top, 8 inches at the bottom, and weighs 240 lbs.; it is not, as is often the case, fortified with wrought-iron rings. The usual keyways for the

removal of stem and shoe are parallel. To fasten on the head to the stem, the latter is let down through the guideholes and the socket of the head is placed beneath it. The stem is then lifted and dropped, and, if necessary, driven on with a sledge-hammer. Then the stem and head are lifted, and dropped several times on to a piece of timber until wedged firmly together. A head lasts five years at the Homestake mill.

The shoes are made of white cast-iron. The cylindrical portion is 8 inches high and $8\frac{1}{4}$ inches in diameter, with a tapering shank 5 inches high, $4\frac{1}{2}$ inches in diameter at the base, and $3\frac{1}{2}$ inches at the top. They are chilled for $6\frac{1}{2}$ inches from the face, whilst the remaining $1\frac{1}{2}$ inches and the shank are cast in sand and cooled slowly. They weigh 140 lbs.

To fasten the shoe to the head the shank is surrounded by small wooden wedges tied on with string, the shoe is put in position and the head dropped several times on to the dies, which are protected by a piece of planking laid across them.

At the Golden Star mill, instead of tying on the wooden wedges, a strip of canvas is wound round them and tacked to each wedge, forming a sort of collar. This can be slipped over the shank of a new shoe, saving time and labour on clean-up days when shoes are replaced,

After some time a shoe becomes slightly concave, but on the whole wears more evenly than the die.

At the Homestake mill a shoe lasts two months, crushing 270 tons of rock. It is then worn down to 2 inches from the shank and weighs 40 lbs., corresponding to 37 lbs. of iron worn away for every 100 tons of rock crushed.

At the Caledonia mill, a shoe lasts three months and crushes 300 tons. It is replaced when worn down to 1 inch and weighs 35 lbs., corresponding to a consumption of 35 lbs. of iron for every 100 tons of crushed ore.

Tappets.—Gib tappets are used, secured in the Homestake mill with two keys, in the Caledonia with three; they are made of cast-iron, and weigh 130 lbs. The diameter at the ends is $9\frac{1}{4}$ inches, in the centre (which is 7 inches long) 6 inches. The wearing faces are $2\frac{1}{2}$ inches thick and are reversible: when both become grooved they are planed off in a lathe and replaced. When worn down $1\frac{3}{4}$ inches (about once in three years) they are replaced by new ones. They rarely split; case-hardened tappets and cams were tried, but the surfaces cracked, and they did not answer well, which has been the writer's own experience elsewhere with steel-faced cams and tappets. Solid steel tappets and cams have not

been tried. It takes between 6 and 8 hours to change the tappets and cams of one battery. The order of drop at the Homestake is 1, 3, 5, 2, 4, at the Caledonia 1, 3, 5, 2, 4, and 1, 4, 2, 5, 3. The Caledonia, crushing harder rock, has a higher drop, 12 inches as compared with 9 inches at the Homestake mill, and consequently has to run more slowly, 74 in place of 85 drops per minute.

Cams and Cam-shafts.—The cams are double and are made of tough cast-iron. The hub, which is on the off-side of the stem, is cast thick enough to stand the strain, but not otherwise strengthened. At the Homestake mill the working-face is 2 inches wide and $3\frac{1}{2}$ inches deep, and at the Caledonia mill $2\frac{1}{2}$ inches wide and 2 inches deep. The hub of both is $3\frac{1}{2}$ inches thick, and the web, which strengthens the cam (commencing deep at the hub, and ending thin at the toe), is $9\frac{1}{2}$ inches deep at the centre in the Homestake pattern and $10\frac{1}{2}$ inches deep in the Caledonia, whilst the distance from the centre of the cam to its point in one case is 17 inches and in the other 19 inches. The cams are made of car-wheel iron and last over four years. They are lubricated with axle grease; and to prevent any of this dropping on to the apron-plates, a curtain of canvas is stretched on a frame below them, and catches any grease thrown off while they are in motion.

The cam-shafts are of tough wrought-iron turned in a lathe. They have one key seat. The keys are of steel, and hand-fitted; wrought-iron keys soon loose their shape, while machine-fitted keys get loose very easily. It takes 10 hours to replace a broken cam-shaft, supposing the keys are ready prepared and fitted. As the fitting of each key-seat takes an hour or more, a well appointed mill should have on hand a spare cam-shaft (or more, depending on its size and situation), with the necessary cams and keys ready for use.

The Homestake cam-shafts were formerly made $4\frac{1}{2}$ inches wide and $4\frac{3}{4}$ inches in diameter, and lasted about five years; now they are made stronger, running up to $5\frac{3}{8}$ inches, and have stood ten years. The distance between the centres of the cam-shaft and stem is $5\frac{1}{2}$ inches.

The cam-shaft of the Caledonia is $4\frac{13}{16}$ inches in diameter, and its centre is $6\frac{1}{2}$ inches distant from the centre of the stem.

The cam-shaft pulleys are built up of wood, varying in diameter from 6 feet to $7\frac{1}{2}$ feet. When put in place the shaft is revolved, and the face turned off true.

Crushing Capacity.—The Homestake stamp develops 78,030,000 foot-pounds in 24 hours, crushing 1 ton of ore for every 17,340,000 foot-pounds

developed. The Caledonian stamp develops 90,576,000 foot-pounds in 24 hours, crushing 1 ton of ore for every 27,447,272 foot-pounds. Thus, although the efficiency of the Caledonian stamp is greatest, it crushes less ore. This is accounted for (*a*) by the greater hardness of the rock; (*b*) the greater width of mortar at the discharge (16 inches as compared with 13½ inches); and (*c*) the recess for the plate at the back of the mortar. With a lower discharge a greater crushing capacity would be expected, but the above reasons explain why this is not the case in practice. The smallness of the Caledonian screen (258 as compared with 376 square inches) may be assumed to be counterbalanced by the Caledonia using No. 24 wire against Homestake No. 7 slot (corresponding to No. 30 wire).

Apron-plates, Traps, and Sluice-boxes.—The pulp passing through the screens flows in small waves down the apron-plate, and during the interval between these waves any quicksilver, amalgam, or fine gold passing over the amalgamated surface has a chance of settling and adhering to it. The plate consists of a single sheet of copper, the width of the mortar-discharge, fastened with iron screws to a wooden table. Except at the Deadwood and Golden Terra mills, which have plates 12 feet long, all the other Homestake mills have their apron-plates 10 feet in length, covered with $\frac{3}{16}$ inch copper plate, falling 2 inches to the foot, and discharging into a copper-lined sluice leading to the mercury-trap. The Caledonia apron-plates are 8 feet long, 4 feet 3 inches wide, and the copper plate is $\frac{1}{8}$ inch thick, set at the same grade as the other mills.

The wooden table extends 4 feet beyond the end of the copper plate, narrowing to 4 feet. It has a 1 inch rib down the centre; it is overlaid by two blankets, 5 feet wide and 22 inches long, the upper overlapping the lower one. On these the heavy sands collect; they are washed every half-hour, and they last six months. The pulp from the blankets flows into the mercury-traps, one being placed in the middle of the discharge from each plate.

The plates are of Lake Superior copper, furnished ready for use, and do not require to be annealed; they must be flattened, however, with wooden mallets to make them lie flat and remove any inequalities.

At the Homestake mill they are first scoured with sand-paper, followed by emery cloth, or with tailings rubbed on with a wooden block, 4 inches square, until the face is perfectly bright. If necessary, the sand is moistened with a weak solution of cyanide of potassium, and black spots are often removed with dilute nitric acid. The bright surface of the copper then receives a washing with a strong solution of cyanide of

potassium applied with a soft brush. After two days, the mercury is sprinkled over the plate and rubbed into it with a moist cloth and tailings. When the plate is thoroughly amalgamated it is put in position. More than the usual quantity of mercury is added to the box at first, so that the plate may get into proper condition. This takes from two to four weeks, and to dissolve the copper salts, which stain it during this period, cyanide of potassium or ammonia is added to the battery water.

The mercury-traps save amalgam and mercury not caught on the apron-plates. There are also additional traps at the ends of the sluice-plates outside.

The importance of this simple contrivance is shown by the fact that since their introduction 80 ounces of amalgam and 144 ounces of mercury are recovered at the Homestake 80 stamp-mill monthly, by the inside traps; whilst the outside ones collect 10 to 12 ounces of amalgam and 40 ounces of mercury. They are emptied monthly. At the Caledonia mill the traps are emptied daily (when the plates are dressed) on account of the accumulation of pyrites.

The inside traps at the Homestake mills are wooden boxes, 14 inches long, 17 inches wide, and 24 inches deep, with a copper-lined bottom. They contain three sliding wrought-iron plates, parallel with the short sides of the boxes, set $2\frac{1}{2}$ inches apart. The central partition extends to the bottom, and the two others are 3 inches above it. The pulp flows under the first, over the middle, and under the third.

The outside traps are 48 inches long, 14 inches wide, and 48 inches deep, with three partitions, set $10\frac{1}{2}$ inches apart, reaching from the bottom, to within $1\frac{1}{2}$, 3, and 4 inches below the level of the inlet, the outlet being 6 inches lower. In the middle, between two of the wooden partitions, a sliding wrought-iron plate, $\frac{3}{8}$ inch thick, reaches to within 3 inches of the bottom of each box. The Caledonia traps are smaller, as there is one for each apron-plate.

The sluice-boxes, which are below the inside traps, are simple wooden launders, lined on the bottom with copper plates. At the Homestake mill, they are 8 to 10 feet long, 18 inches broad, and have a fall of 1 inch per foot. The copper plate is $\frac{1}{8}$ inch thick. At the Caledonia mill they are 8 feet long and 8 inches broad, as less pulp passes through them.

Labour.—All the heads of the different departments are responsible to the superintendent. The mill proper is under an experienced foreman, one foreman being sometimes in charge of several mills.

Next comes the millwright, who in large mills sometimes has an

assistant, called the pipe-fitter. The millwright combines the trades of carpenter and machinist, making and replacing new guides, exchanging cams and cam-shafts, fastening loose cams, replacing screens, making and repairing chuck-blocks, exchanging shoes and dies of crushers, and looking after the water connexions, etc.—in fact looking after all the mechanical work about the mill.

The machinist is in charge of the repair-shop and is generally under the millwright, though at the Caledonia mill the foreman takes this duty; any extensive repairs are made at the Homestake shops.

As the mills are driven by steam, each has two enginemen responsible for their firemen.

There is a night watchman generally for each mill, to guard against fire or other accident.

The man who has the immediate charge of the running of the mill is the head amalgamator. He, like all the other heads of departments, is directly under the foreman, and is in turn responsible for his assistants, amalgamators, crusher-men, oilers, feeders, and labourers. In addition to running the mill he has charge of the collection and keeping of the amalgam, and must therefore be not only capable but trustworthy.

The amalgamators feed quicksilver, regulate the water-supply, and look after the running of the battery in general. Quicksilver is fed every half hour with a wooden spoon like a mustard spoon. The quantity used every 24 hours varies from $\frac{1}{4}$ to $\frac{1}{2}$ lb. for each battery according to the character of the ore.* The correct amount is determined by the feel of the amalgam collected on the plates. If hard and crumbly there is danger of its being carried off by the pulp, and more quicksilver must be added. On the other hand, too much quicksilver makes the outside copper plates too soft and slippery, with the risk of liquid amalgam rolling off, while less amalgam collects on the inner ones.

All the quicksilver is added to the mortar in the Homestake mills, and the amalgam is of medium hardness. At the Caledonia mill the aim is, by adding part to the mortar and the rest to the apron-plates, to make the inside amalgam as hard as may be, and to keep the amalgam on the aprons softer than on those of the Homestake mills.

Each management is satisfied with its own method, and, perhaps, the gold of the Caledonia ore being coarser than the Homestake mill may justify the difference of method.

* Fine gold requires more mercury in amalgamating the same weight of gold, and the loss of mercury is liable to be somewhat increased. Great losses may be occasioned by ores containing heavy sulphides or by over-handling.

The loss of quicksilver at the Homestake mills per year per stamp is 5·27 lbs., or 0·0044 lb. per ton of rock crushed. The loss at the Caledonia mills is 7 lbs. per year per stamp, or 0·0011 lb. per ton of stone crushed.* With the harder and more pyritic ore of the Caledonia mine more mercury is liable to be floured per stamp, but owing to the smaller quantity of rock crushed per stamp, less quicksilver is lost per ton.

The Homestake mills use 1 miner's inch of water per battery, and the Caledonia mill $1\frac{1}{4}$ inches.

To set the tappets, which is very necessary to maintain the height of drop constant, whether the shoes be new or worn, the stamps are hung up, the mortar opened, the stamps lifted by an iron bar, and a block of wood, 1 inch higher than the desired drop, is placed between shoe and die. The tappet is then loosened, allowed to fall on the prop, and again keyed fast. As the point of the prop (finger), and the blocks used to support the shoe are both 1 inch higher than the required drop, on removing the block, the stamps being each in turn regulated thus, will have the desired uniformity of drop, while the different levels at which the tappets are keyed to the stem, will indicate how much the shoe and die are worn down.

The crusher-men, in addition to tending the grizzlies, breaking the coarse lumps, and feeding the crushers, have to watch for and take out any pieces of wood or iron found in the ore, and throw them aside.

All small pieces of wood finding their way into the mortar are removed by the amalgamators, but very little ought to escape the notice of the crusher-men and ore-feeder men, who should remove any pieces they notice, from the shoots of the automatic feeders.

The oilers have to keep all working parts of the machinery properly lubricated, and should be especially careful to guard against excess of grease about the battery.

The feeders attend to the regular and uniform feeding of the ore; the height of ore between shoe and die should never be more than 1 inch, and as much less as possible without allowing the stamps to pound.

One or two labourers are generally needed to do extra work, which does not fall into the usual routine.

The shifts in the mills are changed monthly.

Only three more men are required in the Golden Star 120 stamp mill, than are employed in the 80 stamp mill of the Homestake works running

* In most cases dealing with free-milling ore, $\frac{1}{4}$ to $\frac{1}{3}$ oz. per ton of ore milled, or 12 to 15 lbs. per month, is the loss to be expected in, say, a 20 stamp mill.

on the same ore. From this it will be seen that a large number of stamps, whilst greatly increasing production, does not proportionately increase the labour outlay.

Collection of Amalgam, and Dressing the Plates.—The amalgam which has collected on the apron-plates the previous day is removed every morning with the change of night shift. An amalgamator, each with an assistant, has charge of this work. The method at the Golden Star mill will serve to illustrate how this work is done. When the copper plates are to be cleaned the stamps are hung up, the water is turned off, and the splashboard removed, and washed at the head of the apron-plates with a hose. It is then placed at the lower end of the plate, and the hose is turned on the screen and apron to remove any sand collected on them. The copper plate should now be clear and bright, or silver white where the amalgam has collected, though here and there spots may be left on it, which are generally at first a light yellow, but turn darker with exposure to air. The plates may be scraped with a blunt double-edged chisel. Then two men loosen the amalgam with heavy whisk-brushes, beginning at the top and working downwards. When this is done, the amalgam is swept in the opposite direction, and collected at the head of the apron. There it is brushed into an amalgam scoop with a rubber scraper (a sharp-edged piece of belting) and emptied into a small enamelled-iron dish. After this the plates are brightened by brushing them with a whisk-broom, using tailings moistened with dilute cyanide of potassium solution, the men working from the head of the plate downwards. If necessary, a little mercury is sprinkled on to the plate, from a bottle over the neck of which a piece of canvas is stretched and tied. After being cleaned, the plates are smoothed with soft paintbrushes passed transversely over them, beginning at the bottom. This finishes the operation, which lasts 4 hours for 24 plates, or 10 minutes for each battery. The indiscriminate use of acids or alkalies on the plates is strongly to be condemned, as they tend gradually to alter the nature of the copper, and if applied in excess precipitate verdigris in a few days, or form salts of copper, which, becoming gradually converted into oxides, give additional trouble. The whisk-brooms used for brushing are of the ordinary kind, cut short to stiffen them. The brushing should be done in straight lines, commencing at the top, the amalgam being generally brushed back to the top, but sometimes it is removed at the bottom. It should not be brushed towards the centre.

If plates are not run too wet—*i.e.*, with an excess of mercury, the

chances of oxidation are reduced by this method of procedure to a minimum, and a thin coating of amalgam is left over the entire surface, excluding air and preventing verdigris.

The amalgam obtained is contaminated with impurities. To remove these, it is placed in a mortar and diluted with mercury. The amalgamator then adds water, and grinds the amalgam so as to bring all the impurities to the surface. These may be in part washed off (the sands) with a hose, and in part removed with a sponge or wet cloth, which takes up the base-metal amalgam, until the surface of the mercury in the mortar is bright and clear like a mirror. It is then passed through a small strainer, and the residual pasty amalgam is transferred to a piece of linen, and the excess of quicksilver is expressed by wringing. The ball of hard amalgam is locked up in the safe, and kept till the next clean-up. All the sands are returned to the battery, and the quicksilver goes back into stock.

Clean-up.—Twice a month the gold amalgam adhering to the inner copper plates is removed, and any repairs needed about the mill are made. At the Caledonia mill the bi-monthly clean-ups are similar to that at the Homestake mills on the first day of the month. At the last named mills the semi-monthly clean-up is different. On the first day of the month the entire mortar is emptied, and shoes and dies are changed if need be; while on the 15th only the amalgam from the inside plates is removed, and the mercury-traps emptied.

At the Golden Star mill the clean-up on the first day of the month is carried out as follows:—It begins at 7 a.m., the feeding of the battery is stopped a quarter of an hour previously, and the stamps are made to drop slowly, so that at 7 a.m. no more ore may be left in the mortar above the screen-frame. The splashboards are removed, the stamps hung up, the water shut off, and the engine stopped. The mortars on one side of the mill are then opened by removing the curtains, screens, and chuck-blocks. The curtains and screens are first roughly washed by playing the hose over them, and put aside to be more carefully cleaned later. The six chuck-blocks from the batteries on the side of the mill being cleaned, are placed on two apron-plates, at each of which there are four men stationed to remove the amalgam, under the supervision of the head amalgamator. This is done by scraping the copper plates with a chisel when the hard amalgam drops off on to the apron-plate beneath, as much amalgam as possible being removed without exposing the copper; quicksilver is then sprinkled over the plate (to dilute the hard amalgam), then spread evenly over the plate, brightened by scouring with a whisk-broom and tailings,

and finally smoothed with a soft paintbrush. The amalgam that has dropped on to the apron-plate from the three chuck-blocks is collected at the head of the former, and put under lock and key. Thus the chuck-blocks of the entire mill are scraped and cleaned in four sets of six each.

In the meantime, another set of men scrape and wash the rim and flanges of the mortar and collect the amalgam. They also remove the apron-plate amalgam which has accumulated during the previous day. In order to keep the apron-plates soft a little quicksilver is sprinkled over them and evenly distributed with a brush, but they are not dressed till later.

As soon as the amalgam from the apron-plates has been removed, two small platforms are placed across the head of the table, in front of the mortar, for the men to stand on. They then bale out the water still remaining in the bottom, and shovel out the sands above the dies into a heap on the apron-plate (more usually removed in buckets and collected in a tub at the side). These sands are returned to the battery after the dies have been put in place, as they do not contain amalgam. Before the die can be taken out, the stamp has to be raised higher; to do this a block and tackle were formerly used, now a piece of timber is placed crossways on the rests of the splashboard, serving as a fulcrum for an iron bar, with which the head is lifted. It is kept in position by a 4 inches piece of wood on the prop (finger) of the stamp, on which the tappet is let down. The dies are prized up with the bar, lifted out, and roughly cleaned. Those to be exchanged are taken away, and piled up to be carefully scraped and washed in due time. Those that are still fit for use are returned to the mortar, after they have been scrubbed with a scrubbing-brush in a tub.

After the dies have been taken out, the remaining sand is shovelled out and piled in a separate tub to be treated afterwards in the rocker and pan. It is rich in amalgam and contains bits of iron, etc. Any particles of amalgam that may have adhered to the rough sides of the mortar are washed down and added to the sands. The dies are now replaced, new shoes, if required, are put in place on the dies, and the wooden collar slipped over the shank. Then the recesses of the chuck-block, screen-frame, etc., are cleaned in a tub by playing a hose on them, after which they too are put in.

When the chuck-block is in place, and the screens have been scrubbed down with a brush and water, the sands first taken out are shovelled back to fill the bottom of the mortar up to the top of the dies, and the drop of the stamps is next regulated.

If new shoes have to be substituted for old ones, they are fixed on by letting the heads drop on to them, as previously described; the wooden block, 1 inch higher than the drop, is placed across the dies, and each stamp in turn is let down till the head rests on the block, the keys of the tappets are loosened, allowing them to fall on the props, and they are then keyed up again. The screens are then replaced, the apron-plates being dressed in the usual way; any amalgam clinging to the small sluices leading to the traps and to the sluice-boxes is removed, and these are dressed like the apron-plates.

The splashboards are put back in place, some ore is fed into the mortar, the water is turned on, and the stamps of one battery after another are let down from the props, the engine running slowly at first, and, as the last head falls, gradually picking up speed, till the regular beat of the heads reverberates along the line, a sound as familiar as it is inspiriting to dwellers in mining camps.

As the mortars are empty when the mill starts up, care must be taken to regulate the ore-supply accordingly.

In cleaning up a mill all hands have to take part in it, the night shift working 6 hours extra.

This description of the clean-up of a 120 stamp mill shows how it is possible to accomplish in the short space of 7 hours (without outside help) what formerly used to take a day, through the various operations being systematized and worked into one another. When the clean-up of the mill is over and the stamps are once more running, the sands shovelled out from the bottom of the mortars have to be worked up and the amalgam cleaned for retorting.

Two crusher-men are detailed to clean the sands, which are first washed in a rocker. Any coarse bits of iron are picked out and collected in a dish. When the sands have been rocked for sometime, and the hose played on them, the residue remaining in the hopper of the rocker is broken as fine as possible with a wooden mallet. The coarse particles remaining are washed in a coarse screen over the clean-up pan, any amalgam remaining on the screen being picked out and put into the pan. The sands go back to the battery, and the sulphides, etc., which collect on the curtain and riffle of the rocker, are taken out and put into the pan. The settlings in the sluice which conducts the slimes to the waste-flume are shovelled out, and returned to the battery.

To clean the amalgam collected from chuck-blocks, apron-plates, sluices, mortars, shoes, dies, screens, etc., it is charged with water into the clean-up pan (5 feet in diameter, the muller making 30 revolutions

per minute), and from 600 to 700 lbs. of quicksilver is added.* It takes about three hours to clean in the pan all the by-products containing amalgam. When this is all collected and the water above is fairly clear, the muller is raised with block and tackle, and the entire contents of the pan are emptied through the lowest discharge-opening into a square box which overflows into the tailings discharge-box. The muller and the bottom of the pan are well brushed out, with a stream of water flowing in all the time, and the liquid amalgam in the box is drained of water and passed through a strainer. The pasty amalgam is removed and freed of the excess of quicksilver by wringing it in canvas bags under water. The balls of hard amalgam resulting contain about 38 per cent. of gold. The mercury collected below the strainer goes back to the main stock;† that squeezed from the pasty amalgam is first purified by adding some nitric acid, stirring it, and washing with water.

The semi-monthly clean up is much simpler, only the chuck-blocks are taken out and cleaned, replacing dies and shoes if necessary, and cleaning the traps. Their contents go to the pan and are worked with the other products containing amalgam. This clean-up lasts 5 hours.

Once a year the old iron and wood chips collected during the previous 12 months are worked over. The pieces of iron, which are scraped to remove any bits of amalgam adhering to them when they are picked out from the battery-bottoms, are thrown into a heap in the yard, and left to corrode by atmospheric agency. Oxidation is hastened by adding salt to the heap at intervals. The iron at the end of the year having fallen to pieces, is charged with mercury into the pans and its gold extracted. The chips of wood picked out of the mortar are likewise collected in a box, and are once a year burnt in a heap in the yard, and the ashes are collected and amalgamated in the pan. In this way 16 to 18 lbs. of amalgam is saved every year from the two mills of the Homestake Company containing 200 stamps.

Retorting and Melting.—To remove the mercury from the hard amalgam in balls, cylindrical or bulb retorts are used. The cylindrical retort of the Homestake Mill Company is 12 inches in diameter and 3 feet long, holding 1,000 lbs. of amalgam. The usual charge of 500 lbs. is retorted in about 6 hours, using $\frac{1}{4}$ cord of wood. The retort metal (crude bullion) amounts to 38 or 40 per cent. of the amalgam in the

* Under ordinary circumstances in most mills 200 lbs. is sufficient.

† A large mill of, say, 80 stamps will require about 6 flasks in stock; a 10 stamp mill 3.

charge. At the Caledonia mill it is only 33 per cent.; this no doubt is due to the amalgam being less tightly squeezed. At the Deadwood Terra mill it is often only 25 per cent., and this is accounted for by the fine condition of the gold.

The crude-bullion is melted into bars, using, at the Homestake mill, the 1,500 ounce silver mould (5 inches by 5 inches by 11½ inches), or the 700 ounce mould (3½ inches by 4 inches by 9½ inches). The bars are cast 3 to 4 inches thick, and weigh 1,000 to 1,400 ounces. It takes four hours to melt four 1,400 ounce bars, and the crucible lasts for 8 to 12 charges. The moulds must be warmed and smoked inside before pouring. A little borax or bi-carbonate of soda is invariably added to the melt. If iron be present a little nitrate of potassium is used. Bone-ash is used to thicken the slag and make it skim easily. Phosphorus (in quantities not exceeding half an ounce per melt of 1,800 ounces) may be employed to get rid of the copper, and corrosive sublimate added for bullion containing much lead or antimony.

The loss in melting Homestake bullion is 1·5 per cent., and the average composition of the bars is 820 parts of gold, 165 parts of silver, and 15 parts of base metal. The loss on Caledonia bullion is greater, being 7 per cent., as the amalgam is less carefully cleaned. The average composition of the bullion is: gold, 798 parts; silver, 182 parts; and base metal, 20 parts. The bullion is sampled (chipped or drillings taken), weighed, assayed, and shipped.

Geology of the District.

In the neighbourhood of the Homestake group of mines numerous sheets of porphyry (or more properly speaking felsite) are met with, sometimes cutting across the stratification of the country, but more frequently parallel with it. In the northern half of the belt—the Deadwood Terra and De Smet end—the surface was once overlain by felsite which appears to have been injected between the slates and Potsdam formation of the district, and can still be seen capping the ridges between Gold Run and Bobtail Gulch, and Bobtail Gulch and Deadwood Creek.

Mining.

The cost of operations at the Homestake mine for the year ending June 1st, 1888, was as follows:—Mining, 7s.; milling, 3s. 5½d.; total, 10s. 5½d. per ton; leaving a profit of 4s. 11½d. per ton on 15s. 5½d. ore.

The mining costs were sub-divided as follows:—

	£.	d.
Labour	4	6·50
Dead work	0	11·35
Supplies	0	2·76
Powder	0	0·55
Candles	0	0·84
Machinery	0	2·18
Oil	0	0·64
Timber	0	8·63
Wood	0	2·52
Coal	0	0·21
Total	7	0·18

The cost of mining and milling at the Deadwood Terra mine are stated to be at the present time (1892) 5s. 2½d.* Chlorination-works have of recent years been established in Dakota.

Working Results.

Between June, 1887, and June, 1888, the yield in free gold from the Homestake mines represented 15s. 4d. per ton. Assuming that 85 per cent. of the free gold is saved, the ore would in that case run 18s. 0½d. in value in free gold per ton. Its total value varies from £1 0s. 10d. to £2 1s. 8d. per ton, whilst the amount of concentrates does not exceed 3 per cent. Their value, as shown by experiments, is £5 a ton, the average assay of the tailings is estimated at 6s. 3d. per ton.

Two sets of experiments were made in the spring of 1885 on the Homestake and Golden Star tailings, before and after the introduction of mercury-traps. Before they were introduced 1,124 tons of concentrates (blanketings) had been collected in a separate building (the blanket-house), which assayed £7 6s. 6d. per ton. These, panned down, gave 20·5 per cent. of cleaner concentrates, assaying £8 7s. 5d., and a second grade assaying £3 11s. 2d. per ton. The former yielded by pan amalgamation 56·9 per cent. of their total value.

When, in consequence of these tests the mercury-traps were introduced, the loss was reduced; the concentrates then saved assayed £5 15s. 1½d. per ton, and yielded 92 per cent. of their gold in the pans, but the pyrites still assayed £2 11s. 1d. per ton. The concentration by blankets being too expensive, it was given up.

The tailings from the Highland ore average 4s. 2d. per ton, and from the Deadwood Terra 2s. 1d., seldom running above 3s. 1½d. per ton.

In regard to the fineness to which it is necessary to crush the ore, Dr. Goering made a number of tests to find the relation between size and

* *The Engineering and Mining Journal*, (New York), vol. lv., page 338.

assay value of tailings, samples being taken hourly and the sands obtained dried, weighed, and screened through different sieves. The results were :—

Per Cent. in Weight.	Passing Through Screen. No.	Remaining on Screen. No.	Assay Value per Ton. s. d.
94·07	50	—	5 2½
2·53	50	40	8 10½
3·40	40	—	11 6

These results show that the loss in the tailings increases rapidly if the screens be allowed to remain too long in use. Another set of experiments on tailings running 8s. 4d. per ton, screened through a No. 30 screen, showed that 6 per cent., which did not pass through the screen, assayed as high as £1 0s. 11d. per ton. The result proved that the heavy Russian iron screens should be changed fortnightly.

The Caledonia mill crushed from May, 1887, to May, 1888, 73,425 tons of stone, yielding bullion equivalent to a return of 16s. 9d. per ton in free gold. The blanket concentrates (amalgamated raw in pans) yielded pure pyrites assaying £18 15s. per ton, and the tailings from the blankets when panned gave concentrates worth £1 9s. 2d. to £1 17s. 6d. per ton.

The cost of milling at the Caledonia, a 60-stamp mill, in 1887-88 was 3s. 7½d. per ton, one-third chargeable to labour and two-thirds to material ; a low figure, considering the rock is hard, compared with the Homestake ore.

The two striking features of the Dakota practice are the cheapness, simplicity, and effectiveness of the method by which the free gold is extracted, and the waste of sulphides in the tailings.

Mr. Hofman suggests that in view of the fact that the sulphides would appear to average 3 per cent., worth £5 per ton, they might be advantageously dealt with by passing the pulp through *spitz-lütten* to sort out the coarse sands and mineral, the overflow of the *spitz-lütten* going on to a series of *spitz-kasten*, the outflow of which would be waste. The coarse products drawn from the *spitz-lütten* would contain, according to the experiments made in 1885, free gold, which could be recovered by crushing wet in Chilian mills or rolls, and allowing the pulp to flow over amalgamated plates, and then to pass over a separate series of classifiers (*spitz-kasten*), or go back to the main system. The graded pulp obtained by the different *spitz-kasten* would be separated on round tables into concentrates, middlings, or waste. The middling would be re-worked or pumped back to the main system of *spitz-kasten.*

The cost of concentrating the tailings in this manner would probably

not exceed 4s. 2d. per ton of sulphides. The concentrates could be worked by barrel-chlorination, the total cost of which Mr. Hofman states would probably not exceed £1 13s. 4d. per ton of concentrates. He recommends a combination of two systems of furnaces as likely to cheapen the cost of roasting (where wood costs in this district £1 5s. per cord and labour 12s. 6d. to 14s. 7d. a day). The Spence automatic furnace would do the preliminary roasting cheaply, and the revolving-hearth (the Brunton) would effectually dead roast large quantities of ore, the sulphur of which had been nearly all eliminated. The modern modifications of the latter furnace, such as the Pearce turret or Blake circular hearth, are, the writer believes, specially adapted for dealing with fine ore liable to dust.

The Golden Reward Chlorination Works.

The cost of treating gold ores by barrel-chlorination on a large scale, at the Golden Reward mill, Deadwood, Dakota, has been obtained from careful daily records of the ore treated per 24 hours: these were averaged semi-monthly and monthly, to see where improvements and reductions could be made.* For the months of July, August, September, and part of October, 1891, the amount of ore treated and cost of treatment (including all working expenses, inclusive of interest on capital, taxes, insurance, etc., which always vary with the financial management of works of this character) were as follows :—

1891.	July. Tons.	Aug. Tons.	September. Tons.	First Half of October. Tons.
Amount of ore treated ...	1,430	1,194·75	1,512·75	871
Average amount per day ...	46·13	38·54	50·42	54·43
Costs per ton—	s. d.	s. d.	s. d.	s. d.
Milling	6 2½	5 10	6 0	4 11½
Roasting	6 4½	5 7½	5 8½	5 5
Chlorination	7 4	6 11½	6 10½	6 5½
Office salaries	1 8	1 11½	1 6½	1 4
Construction and repairs	1 6	3 10½	0 10½	1 4½
Total cost per ton	23 1	24 3	21 0	19 6½

In the month of August the works were closed down for one week, while building a dust-chamber and flue to the roasting-furnaces, which accounts for the higher cost per ton and lower daily average of ore treated, also for the large item for construction and repairs. It will be seen that the cost of milling is excessive. This was due, principally, to the very

* *The Engineering and Mining Journal* (New York), vol. lv., page 269.

inconvenient arrangement of the mill requiring many more men to handle the ore than would be necessary if the machinery were arranged differently. The roasting was done in Brückner cylinder-furnaces holding 3 tons to a charge; since then, a revolving continuous feed-and-discharge roasting-cylinder, arranged with self-feeding dust-chamber, has been added, which has brought the cost below 4s. 2d. per ton.

The amount of chemicals used in the chlorination and precipitation departments to treat the ore milled in each month, as above, was as follows :—

	July. Consumption		August. Consumption.		September. Consumption.		First Half Oct. Consumption.	
	Total.	Per Ton.	Total.	Per Ton.	Total.	Per Ton.	Total.	Per Ton.
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
Sulphuric acid ...	37,264	26·82	30,044	25·14	38,083	25·17	21,580	24·77
Chloride of lime ...	15,790	11·04	12,160	10·17	15,840	10·47	8·790	10·09
Crude sulphur ...	542	0·37	381	0·31	479	0·31	253	0·29
Iron sulphide ...	1,106	0·77	952	0·79	1,263	0·82	736	0·84
Costs per ton of ore treated—		s. d.		s. d.		s. d.		s. d.
Sulphuric acid ...	—	1 10¾	—	1 10	—	1 10	—	1 9½
Chloride of lime...	—	1 5¾	—	1 4½	—	1 5	—	1 4⅜
Crude sulphur ...	—	0 0¾	—	0 0½	—	0 0¾	—	0 0¾
Iron sulphide ...	—	0 2¼	—	0 2	—	0 2	—	0 2⅛
Total cost of chemicals ...	—	3 7½	—	3 5	—	3 5¾	—	3 4¾

Mr. Rothwell considers it possible to treat gold ores, similar to those treated in the Golden Reward mill, for from 8s. 4d. to 10s. 5d. per ton by barrel-chlorination, with a mill constructed on the latest improved plans for economical work, and having a capacity of 125 to 200 tons per day.

In the eastern and southern districts of the United States, where labour, fuel, and supplies cost about one-half of what they do in the western, the cost of treatment ought to be considerably less—with such a plant.

On some exceptional gold ores that do not require roasting, and that are not adapted to free-milling, it is possible for chlorination to compete favourably with amalgamation as usually practised in the stamp-mill, by saving a higher percentage of gold. Thus, in chlorination the fine or float-gold will be saved, and the coarser particles having been brightened by attrition and chemical action (by sluicing the leached pulp over copper plates), the gold in this condition, could be readily amalgamated and saved.

A general outline of the Dakota process has already been given,* but the following additional details, embodying the latest improvements in the method as described by Mr. Rothwell†, will be of interest.

The mill, roaster, chlorination, and power buildings are erected on level ground, and the main ore-bins, larger crusher, and dryer on benches cut in the hill side. The chief objection to the Plattner process, for a plant of 50 tons or more capacity per 24 hours, is its enormous size, and the length of time that it takes to complete a single operation. The problem, therefore, that the engineer has to solve in attempting to treat low-grade ores which will not concentrate is to find a process that will treat his ores in large quantity expeditiously, cheaply, and with as little interruption as possible. It is claimed that these objects are attained by the arrangements in question.

The ore from the mine comes in cars to the top of the main ore-bin, which has a capacity of about 1,000 tons. The ore from the bin goes over a grizzly (with the bars set $1\frac{1}{2}$ inches apart) which delivers the lumps to a rock-breaker that breaks it to $1\frac{1}{2}$ inches size. The product of the breaker, joining the fines which have passed the grizzly, is fed to a revolving dryer, which has a cylindrical shell built of $\frac{3}{8}$ inch tank steel, 5 feet in diameter and 18 feet long, set on two heavy cast-iron tyres 4 feet from each end. These tyres rest on adjustable flanged rollers; the roller-frames are bolted to a heavy timber framework, to one end of which is fastened a pivotal casting, while the other end is provided with a set of screw-jacks to permit of the inclination of the cylinder being changed.

The cylinder has usually an inclination of $\frac{3}{4}$ to 1 inch per foot, and is revolved by spur-gearing round its exterior with an intermediate gearing of cone-pulleys and friction-clutch, which gives a range of speed of $\frac{3}{4}$ to 2 revolutions per minute. The friction-clutch allows the dryer to be stopped without stopping the mill, and is much simpler than a fast-and-loose pulley and shifting belt. To increase its drying capacity, the cylinder is divided into four longitudinal compartments by double iron-plates $\frac{3}{8}$ inch thick, bolted to angle-irons riveted to the shell, and an X iron in the centre of the cylinder. The compartments have a length of 14 feet, which allows 2 feet of free space at each end of the cylinder.

The furnace is built of masonry well strapped with iron, and with ample air-passages around the fireplace, and in the bridge-wall, through which a large volume of air can be passed and heated before entering the cylinder.

* *Trans. Fed. Inst.*, vol. iv., page 249.

† *The Mineral Industry*, 1892, page 233.

The dust-chamber is similar, but not quite so large as that of the roaster, to be described later. From the dryer the ore passes by gravity to the fine crusher (rock-breaker) where it is crushed to pass through $\frac{3}{4}$ inch mesh, and is then raised by a chain-elevator to the first screen. This is double, with a coarse mesh within a fine mesh, the object being to protect the latter from undue wear. The screen is hexagonal in form, 9 feet long, 4 feet 6 inches outside diameter at one end, and 5 feet 6 inches diameter at the other. The inner screen is 12 inches less in diameter at each end.

The frame of the screen is a cast-iron hub with six radial arms of 1 inch round iron reduced to $\frac{3}{4}$ inch near the ends. Two of these hubs are keyed on the $\frac{3}{4}$ inch shaft, together with the cast-iron headpiece.

The mesh frames are slid in place in grooved cast-iron pieces fastened to the radial arms of the screen-frame. They are interchangeable, and there are always three spare ones on hand.

The screens make $8\frac{1}{2}$ revolutions per minute. The outside casing is made of double thickness, plain timber inside, and groove and tongue outside, with paper between. The doors, one at each side and one at the end, are hinged to the frame of the casing.

The ore that passes the finer mesh goes to an elevator, and is discharged into the storage-bins; the rejected ore goes through a shoot to a large hopper over the coarse rolls (26 by 15 inches, with heavy steel tyres) which are driven by belts running at 90 revolutions per minute.

The ore from the rolls is elevated to the main sizing-screens, of which there are two of the same construction as the one already described, except that they are only 12 feet long, and are driven with separate bevel gearing and clutch to each screen, so that in case of need one can be stopped, and then the whole quantity of ore passed over the other. The fine crushed ore goes to storage-bins and the coarse to fine crushing-rolls, which are the same size as the coarse ones, and is returned again to the screens.

The rolls are fed by automatic feeders, while an exhaust fan draws off the dust, and discharges it into a collector on the top of the ore-bins. The storage-bins are placed along one side of the mill-building, and have a capacity of about 200 tons of crushed ore. A conveyor carries the ore from the bins to the roasting-cylinder. The latter is 36 feet long, 5 feet in diameter, and built of $\frac{1}{3}$ inch plate. It revolves on five tyres which, like the rollers, spur, pinion, and other gear, correspond with those of the drying cylinder, so that it is only necessary to keep one set of extra parts. The shell is lined throughout with fireclay blocks 6 inches thick, and

moulded to fit the circle. It has projecting shelves the whole length of the cylinder, which raise the ore and shower it through the hot oxidizing gases in the upper part of the cylinder. At the end exposed to the fire, the shell is further protected by a specially shaped block which overlaps the end of the plate, and is held in place by a small piece of square bar iron (fastened to the shell) which fits into a groove in the block. The inclination of the cylinder is 14 inches, and it revolves once per minute.

The dust-chamber, which is arranged to feed the dust back into the roasting-cylinder is hopper-shaped on three sides, the bottom being an inclined cast-iron plate projecting about 8 inches into the upper end of the cylinder. The dust carried out of the cylinder settles in this chamber and, as it accumulates, slides down and mixes with the fresh ore. This arrangement does away with the old auxiliary fireplace and the rehandling of the dust, besides giving the ore to the chlorinators in a much more uniform condition than when the dust is mixed with the ore by hand. From the dust-chamber, the gases pass up an inclined flue on the hillside to a chimney 42 inches in diameter and 60 feet high.

The furnace fireplace is constructed with air-channels and openings through which the temperature and working of the cylinder can be regulated and watched. The fire-arch and bridge-wall are so built that the flame is directed into the lower part of the cylinder, and against the ore as it slides down on the lining. The roasted ore is discharged into a hopper, from which it is drawn into cars and spread on the cooling-floor.

To cool the ore, it is spread out thinly on the floor and furrowed. When the surface has partially cooled water is sprinkled over it, and the whole is raked over again; when sufficiently cool, it is sent to an elevator that discharges into hoppers over the chlorination-barrels. These hoppers are made of No. 14 sheet-iron and have a capacity of between 5 and 6 tons each.

The two chlorination-barrels each have a capacity of 5 tons per charge or 35 to 40 tons per 24 hours. The shells are of tank-steel, $\frac{1}{2}$ inch thick, 9 feet long, and 5 feet in diameter inside, whilst the heads are of cast-iron inserted into the end of the shell and bolted through flange and shell. They are $2\frac{1}{2}$ inches thick and heavily ribbed. The trunnions, which are a part of the head, are 12 inches in diameter and 12 inches long where the bearing comes, and 14 inches in diameter and $4\frac{1}{2}$ inches long inside the bearing, making a total length of $16\frac{1}{2}$ inches; they are bored to fit a 3 inches bolt which passes through them and the entire length of the barrel.

Each barrel has two charging-holes (11 inches by 16 inches) oval in shape. The covers are of cast-iron, and when in place are held down by two heavy yokes and four 1½ inches bolts. An eye-bolt is screwed to the centre of each cover to lift it off, and for this purpose a swinging lever is used which holds it out of the way while the barrel is being charged.

The barrels are lined with lead ½ inch in thickness on the heads, and weighing 24 lbs. per square foot, while the shell and other parts exposed to gas or solution are covered with lead ⅜ inch in thickness, and weighing 18 lbs. per square foot. The steel shell is made in one sheet with butt joint and cover-plate, and all the rivet heads are countersunk, so that the inside is perfectly smooth. The lead lining is bolted on with flat-headed lead-covered bolts, which prevents any solution getting between the lining and the shell. The barrel is driven with spur-gearing encircling the shell by a pinion and friction-wheel, which can be thrown in to or out of gear, and a brake is also arranged so that the barrel can be stopped at any point in its revolution.

The supporting diaphragm for the filter is placed so that it will assume a horizontal position when the charging-holes are in place for charging. This diaphragm has an area of nearly 30 square feet, being 8 feet 2 inches long by 3 feet 8 inches wide, and is put into the barrel in the following manner :—Two strips of wood, 2½ inches thick on one edge and 1½ inches on the other, 6 inches wide, and the length of the barrel inside, are bolted to the shell through holes left for that purpose. Below these strips is built a lining of wooden staves, 1½ inches thick and 5 inches wide, on which are placed the supporting segments 3 inches thick. One is placed against each head, and five others spaced equidistant between, which brings them nearly 13 inches apart. On the top of the segments the corrugated plates are laid lengthways with the barrel. These plates are of wood, 2 inches thick, four of which in width form the filtering surface. Each plate is corrugated lengthways with grooves $\frac{5}{16}$ inch wide, ⅜ inch deep, and ⅜ inch between each groove, while ⅜ inch holes are bored through the plate 3 inches from either edge, and every 5 inches lengthways cross-grooves are cut to intersect these holes. On the plates is spread the filter and asbestos cloth, woven a little finer than the ordinary gunny sack. Over this is placed an open wood grating of 1 inch by 1½ inches slats, with openings 3½ inches by 9 inches. This grating and the whole filter are held in place by five heavy brace-pieces 3 inches in thickness, the ends of which slip under the wooden strips bolted to the shell ; small spacing-pieces are placed between the braces, which prevent their coming out or shifting out of place when the barrel is revolving.

All the woodwork of the interior of the barrel is previously boiled in tar and asphalt until saturated, which prevents it absorbing solution and lengthens its life.

Below the filter at each end of the barrel are the valves through which the solution is drawn to the slime-filters or the storage-tanks.

Above the filter, and between the charging holes and the ends of the shell are the valves and connexions through which the wash-water is pumped for leaching.

On one side of the barrel is placed a large storage-tank built of plank and timber, and lined with sheet lead weighing 6 lbs. per square foot, in the bottom of which is an ordinary quartz filter.

Directly below the solution-valves are the slime-filters which are connected with one another by heavy 2 inches acid-proof hose with special connexions. The slime-filters are cast-iron cylinders, flanged on each end, 30 inches in diameter by 18 inches in length, with cast-iron covers, having inlet and outlet-pipes bolted on. The cylinders are lined with sheet lead weighing 8 lbs. per square foot, and have filters similar to those in the barrel, except that above the asbestos cloth a quantity of quartz sand of different degrees of fineness is spread to a depth of 6 inches; and on the top of this there is a second asbestos cloth, arranged to be lifted out and washed when the fine slime and sand have accumulated to an extent which would retard filtration.

In front of the slime-filters are the precipitating-tanks, two for each barrel. They are placed so that the top of the tank is a little below the outlet of the filter. They are 6 feet 6 inches in diameter by 10 feet 6 inches in height, and made of $\frac{3}{8}$ inch tank-steel, with heavy cast-iron flange and cover on the top end, and at the bottom a $\frac{3}{4}$ inch circular plate is flanged and riveted on the outside of the shell, all rivets being countersunk so as to present a smooth surface for the lead lining to rest upon. A 3 inches bolt passes through the centre of the cover and the bottom and a casting placed as a large washer. The cover has a manhole and three 2 inches holes, from two of which lead-pipes extend nearly to the bottom of the tank. The tank is lined throughout with sheet lead weighing 8 lbs. per square foot. There is a 2 inches hole 9 inches from the centre of the bottom-plate, and another 2 inches hole on the side, just above the top edge of the flange of the bottom-plate. The tanks are supported in position on four heavy iron brackets riveted to the shell about 4 feet above the bottom, which rests on a timber-frame, bringing the bottom of the tank about 4 feet 6 inches above the floor.

Between each set of tanks is placed a filter-press with twelve chambers,

19 inches square and $\frac{3}{4}$ inch distance-frames. The press has a filtering area of 57 square feet. Two heavy lead pipes from each tank, one from the hole in the bottom and the other from the hole in the side, lead to the press, and each pipe has an acid-proof valve close to the tank.

The generators, in which the gas for precipitating is generated, are placed on the floor at the top of the precipitating-tanks. They are plain cast-iron cylinders, of the same size as the slime-filters. The one for generating the sulphurous acid gas is not lined, but has a cast-iron tray and delagrating-plate. The other for generating the sulphuretted hydrogen gas is composed of two cylinders, one above the other, with a plate having a 3 inches hole in the centre between them. The cover of the upper cylinder has a hand-hole 8 inches in diameter in the centre, and a 2 inches hole near one side. The lower cylinder-cover has a $2\frac{1}{2}$ inches hole in the centre, and a small hole tapped for a 1 inch pipe near the top on one side. Both cylinders and covers are lined inside with lead, and a $2\frac{1}{2}$ inches lead pipe is put in the 3 inches hole in the plate that separates the two cylinders. One end is burnt on to the lead lining on each side of the plate, and the other reaches to within $1\frac{1}{2}$ inches of the bottom of the lower cylinder.

The method of operation is as follows :—The ore from the cooling-floor is elevated to the hoppers over the barrels, and from there is charged into the barrel in which the requisite quantity of water and sulphuric acid for the charge have been put. After the ore, mixed with chloride of lime, has been added, the cover is put on and screwed down tight. The friction-gear is engaged, and the barrel allowed to revolve for $1\frac{1}{2}$ to 2 hours, at the end of which time it is stopped, with the charging-holes so placed that the filter is horizontal. The connexions are then made from the pressure-pump to the barrel with the slime-filter and precipitating-tanks. The pump is started and water forced into the barrel, the pressure being seldom above 40 lbs. per square inch. The first water entering absorbs any free chlorine gas left in the barrel after chlorination, and the gold solution is delivered quite clear into the precipitating-tank. Each precipitating-tank has a capacity equal to the solution from two charges. The storage-tank under the barrels is used in the event of there being more solution than will fill the precipitating-tank, and when the filter-cloth wears out and lets the sand through, the solution that accumulates here is afterwards drawn into the precipitating-tank and the gold precipitated.

After leaching the charge in the barrel, it is emptied into a tank below the slime-filter floor, and thence sluiced to the tailings-dump. The asbestos filter-cloth is washed, so as to free it from any sand that may clog

the interstices of the cloth by directing a stream of water under pressure through a small nozzle against every part of it, the water thus put in is discharged by revolving the barrel, which also washes out any tailings that still remain. The life of a filter-cloth is between 50 and 60 charges.

As soon as the precipitating-tank is full of solution, the sulphurous acid gas generator is started and the gas is forced through one of the pipes leading to the bottom of the tank. The gas is generated from sulphur burnt on a tray in the generator, with a current of air forced in at the bottom and deflected over the surface of the burning mass. The excess of air carries the gas through the pipe into the solution, in which it is rapidly absorbed, the air acting as an agitator. When sufficient gas has been passed into the solution to convert all the free chlorine gas into hydrochloric acid, the sulphuretted hydrogen gas is generated from sulphide of iron and dilute sulphuric acid, the acid solution being poured into the lower cylinder of the generator, upon the sulphide of iron placed on a perforated lead plate close to the bottom of the upper cylinder. Air-pressure is turned into the lower cylinder through the hole in the side, driving the acid solution up through the $2\frac{1}{2}$ inches lead pipe into the upper cylinder, where it comes into contact with the sulphide of iron. The air is also allowed to pass through the generator to the bottom of the precipitating-tank, where it acts as an agitator, and collects the precipitated sulphide in a flocculent form. As soon as the precipitation is complete, the air is shut off and allowed to escape from the lower cylinder, when the acid solution recedes from the sulphide of iron, and the gas ceases to be generated. The acid solution is used over again, until saturated with sulphide of iron.

The precipitate in the tank is allowed to stand for about an hour, in which time most of the sulphide of gold has settled to the bottom. The valve on the pipe leading to the filter-press from the side of the tank is then opened, and the supernatant liquor is allowed to pass through the press, in which any sulphide of gold still in suspension is collected. When filtration becomes slow, air-pressure is turned into the tank and the liquor is forced through the press. After four precipitations the precipitate which has collected in the bottom of the tank is forced into the press, through the pipe in the bottom of the tank. To compress the sulphide-cake in the press, air is allowed to blow through till the filtrate stops coming. The press is then opened, the sulphide-cake is removed, dried, roasted, and put away till the clean-up is made, when the accumulation from 15 days' run is put in a crucible with borax, nitre, and a little quartz sand, fused, and cast in a mould.

The amount of precipitants used to precipitate a tank of 2,500 gallons

of solution is—sulphur, 2 lbs.; sulphide of iron, 4 to 5 lbs.; sulphuric acid, 16 lbs.; and 9 gallons of water.

The capacity of the works is 75 tons per day.

Power for the whole of the works is furnished by a 125 horse-power engine and two 75 horse-power tubular boilers, which are largely in excess of the amount required. A small air-compressor and steam pump take steam from the boilers.

The number of men needed to operate the plant is 30, including the chemist and superintendent.

For power, 6 to 7 cords of wood or 4 tons of bituminous coal are required per day; for roasting, 5 to 6 cords of wood are used per day. The chemicals required are—chloride of lime, 8 lbs. per ton chlorinated, and sulphuric acid, 15 lbs. per ton.

The wear and tear of plant, oil, etc., is estimated at between 1s. 1d. and 1s. 3d. per ton of ore treated.

The buildings to cover the plant are—coarse-crusher and dryer, 34 feet by 38 feet; mill, 38 feet by 40 feet; roaster, 40 feet by 62 feet; chlorination-shed, 36 feet by 54 feet; engine and boiler-house, 46 feet by 40 feet. Space is allowed for one more set of rolls, a second roasting cylinder, and two extra barrels, which would nearly double the capacity of the works.

The Golden Reward mill has paid £12,500 in dividends during the year 1892, and is now treating between 90 and 100 tons per day with 4 barrels at work.

Practice in the Otago District, New Zealand.*

The province of Otago, in the Middle Island, was the first gold-milling district opened in this colony. Its history dates from the famous discovery of auriferous gravel in Gabriel's Gully in June, 1861. Though it has ever since continued to be more of an alluvial field than a quartz-mining camp the last few years have witnessed considerable progress in the development of quartz-reefing, the importance of which is becoming better recognized. For the year ending March 31st, 1892, the yield of gold amounted to 105,531 ounces, worth £423,527. The milling practice bears the impress of the system followed in the older centre of Victoria, but altered conditions have induced certain variations which give it undoubted characteristics of its own.

* The writer is indebted for the particulars given of this and the two following districts to papers by Mr. T. A. Rickard on "Variations in the Milling of Gold-ores." *The Engineering and Mining Journal* (New York), vol. lv., pages 78, 101, 222, 247, 389, 416, 534, and 560.

The three mills, statistics of which are given in the following comparative table, are fairly typical. The Phœnix battery at Skippers is one of the best known in New Zealand, and nestles at the base of the snow-capped range of the Southern Alps. The Premier mill at Macetown is an old one, shortly to be replaced by a larger plant. The Reliance is at Xenthorn, a mining township, which, though of comparatively recent origin, has already passed through many vicissitudes of chequered prosperity.

Name of Mill.	Stamps.				Depth of Discharge.	Capacity of Stamps per Day.	Capacity of Mill per Day.	Grating Used.	Holes per Square Inch.	Concentrates.	Bullion.	Retort.	Wear of Gratings.	Loss of Mercury per Ton of Ore Treated.	Consumption of Water per Minute.
	No. of.	Height of Fall	Weight.	Drops per Minute.											
		Ins.	Lbs.		Ins.	Tons.	Tons.		No.	%	÷1000	%	Days.	Dwts	Gals.
Phœnix ...	30	7½	800	78	3½	1·4	40	Steel cloth	140	Not saved	930	45	6	8·4	4
Premier ...	10	7	750	77	6½	1·2	12		180		949	34	8	7	3½
Reliance	10	7½	850	75	2½	2·0	20		200		850	35	12	5	5

The Phœnix mill, at the head of Shotover river, was one of the pioneers in the utilization of electricity for the transmission of power. The plant was erected in 1884, and consists of two Pelton water-wheels, working under a head of 180 feet, which drive two Brush dynamos, connected by two No. 8 B.W.G. copper wires, forming a line (nearly 3 miles long) to a Victoria motor, which in turn supplies power to run 30 stamps, together with a rock-breaker, and lights the buildings and plant.

Two kinds of grating or screen are used, wire-cloth being chiefly employed, but when the supply runs short the ordinary round punched iron is substituted. The holes in the two cases are of similar size, but the number per square inch is 324 in the one case and only 140 in the other.

The pyrites in the ore has been proved to be of very low grade, and no concentration is therefore attempted.

The loss of mercury was 90 lbs. on a crushing of 3,107 tons of ore. The gold saving is effected by mortar-boxes and blanket-tables, the residues from the one and the washings from the other both undergoing supplementary treatment in an amalgamating-barrel.

The method of milling is extremely simple: the ore passes through a rock-breaker and is fed automatically into the battery, on the principle adopted at Clunes, which has been already described.

The mortar-box has a depth of 9½ inches from the lip to the bottom, divided as follows;—From bottom of grating to top of die 2½ inches;

thickness of die 4 inches, and false bottom 3 inches. This false bottom consists of two sets of four bars, which are placed under the dies, and are packed in between with sand. Each bar is 3 inches square, and has a length of 2 feet 5 inches. The space between the outer bars and the side of the mortar-box is $3\frac{1}{2}$ inches, the width of the coffer being $15\frac{1}{2}$ inches. The arrangement is an expedient for surmounting the difficulties presented by a mortar, whose shape is unsuited to the nature of the milling required by the ore. Before starting the mill the coarse sand from the previous clean-up is packed between and around the dies.

The order of drop of the stamps is 3, 5, 1, 4, 2. No mercury is added to the ore when in the mortar-box, the gold being caught by the action of gravity alone.

On leaving the mortar-box the pulp has 3 drops, making 18 inches in all, before it reaches the uppermost blanket. This fall serves the purpose of spreading the material. There are no amalgamating-tables, and the pulp passes immediately over the blanket-strakes. These last have a length of 18 feet and a width of 6 feet, subdivided into 4 runs. The gradient is $\frac{3}{8}$ inch per foot, and about 4 gallons of water is supplied to each stamp per minute, delivered to each set of 15 stamps (3 batteries) by two $1\frac{1}{2}$ inches pipes under a head of 25 feet.

The blankets are freed of the gold and heavy sand that they collect by vigorous rinsing in a tub of water. The top row of the four runs of blankets is washed every hour, and the lower ones are washed at longer intervals, depending on the richness of the ore.

The blanketings or residues from the washing are removed from the tub when a certain quantity has accumulated, and are conveyed in buckets to a barrel (5 feet by 4 feet), having a capacity of 120 gallons (equal to about 1 ton of blanketings). In running 25 to 30 stamps on average 15 dwts. ore, a sufficient supply for a barrel is obtained each day. The water in the barrel is used cold, and 75 lbs. of mercury is added to it. Experiments are being made with sulphate of copper. The barrel turns at a speed of 20 revolutions per minute. When amalgamation is thought to be completed, usually after 24 hours, the material is emptied into a vat; thence it is slowly fed by a running stream of water to a shaking-table of the Rittinger type, 8 feet in length and 2 feet wide. Below the table there are a few pieces of copper plate, but these do not serve much purpose. The collection of amalgam from the contents of the barrel by the shaking-table occupies $2\frac{1}{2}$ to 4 hours, depending on the rate of feeding, and this varies with the heaviness of the pulp.

The amalgam, pyrites, heavy sand, etc., thus roughly concentrated are

placed in enamelled iron buckets to be further washed by hand in a pan. This latter has the shape of the ordinary gold pan, but is made of copper, and its surface having been amalgamated by frequent use, it readily collects amalgam.

Mr. Rickard states that on examining the tailings below the shaking-table he found that they contained a large amount of floured quicksilver. The average loss of 8·4 dwts. of mercury per ton of ore is above the ordinary, and is no doubt due to the high speed of running the barrel.

At the monthly clean-up the battery-residues are roughly screened on a riddle, and the larger bits of quartz removed, previous to adding the residue to the blanketings for treatment in the barrel.

The friable character of the mill-stuff makes the wire gratings used preferable to those of punched iron. It is more easily placed in position on the screen-frame, has a much longer time of wear, and much greater area of discharge. The punched iron gives finer crushing at first, before the burr is worn off, but afterwards becomes easily choked up.

The short life of the gratings, a week for the wire, and slightly less for the punched iron, is not due to anything in the ore itself, which is a comparatively clean quartz, but to the fragments of wood (from mine timbers) which to a quite unusual extent find their way into the mill-stuff. They choke up the gratings which, by the pressure of the water and pulp thus held back, are burst. This is notably the case with the punched iron, which discloses lines of weakness along the vertical divisions made by the press employed in its manufacture. The wire cloth, No. 18 mesh, costs 9s. per yard. It is sold in pieces 30 yards long by 2 yards wide. The grating is cut out of this to the size of 2 feet 6 inches by 10 inches. On the other hand, 120 punched Russian iron gratings cost £17 or 2s. 10d. each, as against 2s. 2d. each for the wire cloth. The expenditure under this head amounts to £5 per month for the 30 stamps.

In place of blankets green baize is used, costing 3s. per yard, and has a time of service varying from three to four months. The expenditure per month is equal to £7 for the entire mill. Those least worn are always placed in the first row. The washing is done by boys who receive 7s. 6d. per shift. One stout lad will do the work demanded by three batteries, but cannot manage the washing of the blankets of four batteries, *i.e.*, 20 stamps.

The ore is essentially of the free-milling type, and is broken from a large quartz-lode traversing schists. The quartz often has a laminated or ribbon structure which enables it it to be readily broken. Inclusions (horses) of country rock are common, and the stone only carries $\frac{1}{2}$ to $\frac{3}{4}$ per cent. of pyrites.

Concentrating tests have shown that the highest grade of concentrates obtained contains only 10 dwts. of gold, giving a value too low for treatment in this locality, and on the small scale required by the circumstances of the case. The gold occurs free. Ore containing more than the usual percentage of pyrites is generally below the average grade,* the gold is not therefore notably associated with the pyrites. It is usually coarse, and often visible. Quartz, which in the stopes does not on examination show occasional specks of gold, is generally of low tenor. On being crushed, the matrix of quartz readily separates from the particles of gold. It might be anticipated that the coarsest gold would be caught in the mortar-box, and that obtained from the first row of blankets would be less fine than that caught on the bottom one, and such is the case. Pieces are found in the battery very coarse indeed, weighing occasionally 5 to 8 dwts.

The mill is lighted by arc lights, and on examining the blanket-strakes by the aid of this powerful light the yellow gleam of the gold particles can be observed scattered over the green baize.

A clean-up shows the distribution of the gold-saving:—230 tons yielded 691 ounces of amalgam, of which 270 ounces came from the blanketings and 421 ounces from the mortar-box residues. On retorting, 313 ounces of bullion resulted, which on melting was reduced to 301 ounces 4 dwts., worth £3 19s. 3d. per ounce.

The most striking features of the milling-practice of the district are that, as a rule, no mercury is used in the mortar, its use being confined to the after treatment, the gold-saving being effected by gravity alone.

It will be allowed that the simpler mill treatment is, the better, because it is usually also cheaper. Another milling axiom is that the treatment should be adapted to the nature of the ore. Here, if the methods employed are elementary, the character of the mill-stuff is no less remarkably simple. Whether the mill succeeds in the extraction of a proper percentage of the value of the gold in the ore is the only question, to be decided. Repeated assays of the tailings from the Phœnix mill prove that excellent work is being done. The composition and character of the ore justify the entire replacement of the ordinary copper plates by blankets, and the successful extraction confirms this method.

In milling, as in mining, one is apt to generalize somewhat hastily, and the good work done by this mill has made the manager an enthusiastic advocate of blankets and an equally pronounced enemy of amalgamating-plates. In an experiment carried out at this mill, two 5 stamp

* In this respect it differs from the ore of many other localities.

batteries were supplied with 80 tons each of the same kind of ore. No. 1 battery was provided with mercury inside the box, with copper amalgamating-tables outside and mercury-wells, and, finally, two rows of blankets; No. 2 battery was supplied with no mercury, and was supplemented by blankets alone. The result of the test showed that 8 ounces (or 2 dwts. per ton) more were obtained by No. 2 than by No. 1 battery.

In condemning copper plates the manager equally objects to the use of mercury in the rest of the mill, and would confine its employment to the final collection of the gold in the blanket-washings. As a case in point, and to confirm the correctness of his ideas, he instanced the Invincible mill on the other side of the same range of mountains where the gold-saving was done by the mercury in the battery itself by wells, amalgamating-tables, and, lastly, by blankets. On ceasing to add mercury to the ore in the mortar-box, it was found that more gold was saved.

The two instances at the Phœnix and Invincible mills merit careful examination. It was scarcely surprising that the addition of mercury to the ore at the Invincible mill, when in the mortar-box, did not improve the gold-saving, nay, that it caused a loss, for the mortar-boxes are merely square iron boxes in no way adapted to the particular work required of them.

The explanation of these abnormal results is to be found in the fact that the mortars were not designed of a shape adapting them to inside amalgamation, and there was no opportunity given to the amalgam to collect out of reach of the falling stamps, but, on the contrary, the quicksilver added was subject to a violent agitation, which caused it to become floured. The particles of mercury thus broken up are readily carried off by the water, and, escaping with the tailings, take with them a certain quantity of gold.

At the Phœnix mill the experiment quoted is vitiated in a similar way, as a mortar-box does not become a successful amalgamating-machine by the mere addition of quicksilver. The batteries of this mill are rectangular in section, with vertical ends and sides, and are in no way adapted to inside amalgamation.

To make a fair comparison between the effectiveness of amalgamation, as against blanket-saving, it is necessary to have the two types of batteries—one roomy and of particular shape, the other narrow and rectangular—kept in view when considering their suitability for the two methods of milling; but there is no suggestion here intended that blankets could be advantageously replaced by copper plates at the Phœnix

mill. Different ores require different modes of treatment, and if blankets will arrest the gold, it is obviously not advisable to use expensive quicksilver or to go to the trouble of copper plates.

The mode of milling at the Phœnix mill is of the utmost simplicity, but it is suited to the ore whose gold contents it is designed to extract.

The Premier mill at Macetown is a much smaller plant, but is engaged in the treatment of a somewhat similar ore by slightly modified methods. The mill possesses 10 heads, weighing 700 lbs. each, and the speed varies from 75 to 80 drops per minute. The height of drop has a maximum of 9 inches and a minimum of 6 inches, according to the hardness of the mill-stuff.

The issue or depth of discharge averages $6\frac{1}{2}$ inches, from 6 inches when the dies are new to 7 inches when worn down. The depth is regulated by the insertion of a blind or blank piece of sheet-iron inside the screen-frame, which increases the issue at the start when fresh dies have been placed in position. As the dies wear down a smaller piece is inserted, and finally the full depth of the screen is utilized. The grating is of round punched iron, 180 holes per square inch. The bullion is 949 fine, and the amalgam yields on retorting 30 to 38 per cent. The gold saving is done by the mortar-box, to which mercury is added, by copper plates on the tables outside, by wells, and finally by blankets supplemented by a berdan pan.

There are no rock-breakers in use, the feeding of the batteries being done by hand. The mortars of the two 5 head batteries are of different patterns. One is more roomy than the other, and therefore discharges the pulp more slowly.

Seeing that amalgamation inside the box is desired, the millman is right in preferring the wider coffer, since it gives more shelter to the particles of gold and mercury. On examination it was found, as might have been expected, that the pulp issuing from the wider mortar was finer than that from the other, although the same kind of screen was used in both.

Contrary to the usual practice, the blankets precede the copper plates; the pulp has a drop of 22 inches to spread it before it falls on to the first row of blankets. These are 12 feet long and 4 feet 3 inches wide, divided by three longitudinal partitions. They slope $1\frac{1}{2}$ inches per foot. The blankets succeed each other in three equal lengths. The first or top row is washed every hour, the second every alternate hour, and the third every third hour. Then follow the copper amalgamating-tables, 9 feet long by 4 feet wide. The total length is subdivided by

five wells, one each at the top and bottom, and three others at equal distances between the plates. Of the five only three are supplied with mercury. They are 3 inches wide and only ½ inch deep.

The residues from the blankets are shovelled from one tub into a second, from which they are fed by a running stream of water into a berdan pan 4 feet in diameter. Instead of the ordinary ball a suspended muller, called the drag, placed at one side of the pan, does the grinding: this modification keeps the grinding and amalgamation separate, thereby preventing unnecessary flouring of the mercury.

A copper plate, 4 feet 8 inches by 2 feet, is placed below the berdan pan with the view of arresting any amalgam escaping in the slimes. At the lower end of the plate there is also a mercury well. The berdan pan makes one revolution for every three drops of a stamp; that is, 25 revolutions per minute, when the average speed of 75 drops per minute is being maintained.

Of the total quantity of amalgam obtained, 60 per cent. is found inside the mortar, 33 per cent. in the blanket-washings, and the copper-tables save the remaining 7 per cent. It was found that by using copper plates below the blankets, instead of a fourth row of blankets, about 5 per cent. more of amalgam was obtained. This is of interest, as proving (what might be inferred) that copper plates are particularly adapted to arresting fine gold, just as blankets are suited to catching it when coarse.

It will be noticed that at none of the mills is there any effort made to concentrate the pyrites. As a rule (the Phœnix ore being a notable exception) the pyrites of the Otago lodes yield a very good grade of concentrates. There is, however, no chlorination or smelting-plant in the province, and any concentrates obtained are shipped to Australia for treatment, at a cost and delay proportionate to the distance. That fact goes far to explain the neglect of this part of the milling.

Both the Phœnix and Premier lodes carry ore, the gold of which is coarse and free; this explains the comparatively crude and very simple method of treatment. Under such favourable conditions blankets are very effectual contrivances for arresting the gold. This system of gold-saving is of very early origin: it was used in America before the discovery of gold in California. The mining districts of the Sierra Nevada borrowed it from the miners of Georgia, and they in turn owed it to those of Verospotak and Nagyag in Hungary. It came back eastward when the discovery of the Gregory diggings started the mining industry of Colorado. It was derived by the millmen of Otago from the mills of Clunes, which, like those of the United States, borrowed it from Europe.

Blankets mark the infancy of milling and belong to the gossan stage of mining: they can only survive those changes in the ore which obtain with increased depth when that ore remains, as rarely happens, unaccompanied by much pyrites, and that pyrites not associated much with the gold.

The Premier mill uses less water than the Phœnix, because the blankets of the latter have less gradient and larger surface. At the Premier mill, mercury is added to the battery-box, while at the Phœnix this is not done. The latter is probably the more correct practice. The gold is coarse and free, and other things being equal, when a large percentage can be arrested by the blankets, it is probable that the still coarser particles which remain inside the battery would be caught there by gravity without using mercury. In both mills the final extraction of the gold from the blanketings is roundabout and clumsy.

In conclusion, while it may appear that the mills of Otago have but little that can be advantageously imitated by those of Colorado or California, because they are adapted to the treatment of an ore of a very simple character, yet the examination of their modes of working may be of value to the millman in causing him to ponder over the why and wherefore of many parts of his own practice, whose advantages he is too ready to accept without always questioning and considering its actual merits.

Practice at Ballarat, Victoria.

In the year 1891, the output of the Ballarat district amounted to 202,704 ounces 1 dwt. 12 grains, of which 127,971 ounces 8 dwts. 9 grains were derived from quartz mining, whilst the remainder was derived from alluvial sources. Dividends amounting to £222,839 15s. were paid during the same period. The quartz yielded on the average at the rate of 8 dwts. 1 grain per ton, while the pyrites (concentrates) contained 2 ounces 3 dwts. 2 grains of gold per ton. The value of the gold ranged from £3 17s. 6d. to £4 3s. per ounce.

The table on the opposite page illustrates the chief features of several of the leading mills.

The Star of the East at Sebastapol is the most productive quartz mine in Victoria, it produced 34,092 ounces of gold, and paid £79,200 in dividends in 1891. The company own two mills, in the newer one the centre stamp drops ½ inch less than the other four, which is said to produce a better splash of the pulp against the grating. The order of drop is 5, 4, 3, 2, 1. When the gold in the ore is found to be unusually

fine the dies are allowed to wear down more than the ordinary maximum of 4 inches, so as to obtain as deep a discharge as possible, the depth at starting being 2 inches.

The gratings carry 200 round punched holes per square inch and last twelve days, but sometimes, when the gold in the ore is very finely divided, gratings with 270 holes per square inch take their place; these wear three to five days.

	Name of Mill.				
	Star of the East (New Mill).	Star of the East (Old Mill).	Britannia United.	New Normanby.	North Cornish.
Stamps—					
Number	60	20	40	20	50
Weight (lbs.)	1,008	784	1,050	784	784
Number of drops per minute	73	75	60	60	72
Height of drop (inches)	$8\frac{1}{4}$	$8\frac{1}{4}$	8	7	8
Average depth of discharge (inches) ...	3	3	$1\frac{1}{2}$	$4\frac{1}{2}$	$1\frac{1}{2}$
Crushing capacity per stamp (tons)	2	1·5	2·1	2	1·8
Crushing capacity of mill (tons)	120	30	84	40	90
Description of grating	Round punched			Russia iron.	
Fineness of grating, holes per square inch	200	150	120	120	160
Percentage of concentrates (per cent.) ...	$3\frac{1}{2}$	$3\frac{1}{2}$	1	—	$2\frac{1}{2}$
	oz. dwts	oz. dwts.	oz. dwts.		oz. dwts.
Tenor of concentrates per ton	3 9	3 9	1 7	—	5 13
Fineness of bullion (per 1,000)	970	970	978	965	935
Retort percentage	46	46	50	70	35
Wear of gratings (days)	10	12	14	12	6
Consumption of mercury per ton of ore (dwts.)	5·7	5·7	2·6	5·6	9·4
Consumption of water per stamp per minute (gallons)	$7\frac{1}{2}$	$7\frac{1}{2}$	5	5	$2\frac{1}{2}$

As a rule the gold is fairly coarse, there being, however, a marked difference in the product of the two lodes worked in the mine. The amalgam from No. 2 shaft, the ore of which averages 13 dwts. per ton, retorts 45 per cent., while the ore from No. 1, which yields 17 dwts. per ton, retorts 48 per cent., the gold being coarser.

In the new mill, the gold is partly caught in the mortar-box by the introduction of a tablespoonful (4 ounces) of mercury every 2 hours. Outside the tables are covered with plain copper plates, and are given a grade of $\frac{7}{8}$ inch per foot. At their lower end there are two drop wells and one shallow well, which catch but little gold when the plates above them are in good order: they simply serve to arrest any mercury escaping from above. Below are blankets. The blanketings—the residues from the regular washing of the blankets—are stacked, lime is added, and they are allowed to stand two days, after which they are re-introduced into the battery with the usual ore-feed. Below the blanket-strakes are eight

ordinary shaking-tables. The pulp, escaping from these, passes over ties or straight sluices. Three berdan pans are used for grinding the skimmings from the wells.

In the old mill, amalgamation is effected in the mortar, and outside on copper plates, which are given an inclination of $\frac{3}{4}$ inch per foot. They are preceded, however, by two, and followed by three wells. Then come two strips of blanket, 8 feet in length, with a gradient of $1\frac{1}{4}$ inches per foot, succeeded by four shaking-tables, one to each battery. Two berdan pans are used for treating the skimmings: the grinding and amalgamation being commenced with a ball and finished with a drag.

The old mill is stated to be doing the better work of the two, the main distinction between it and the new one being that the copper plates of the latter do that part of the gold-saving which, in the old 20 stamp battery, is accomplished by the upper wells. In both cases most of the gold is caught in the mortar-box by gravity, assisted by the free use of mercury; whilst the larger portion of what escapes the battery is caught by the copper plates in the new mill, and in the old plant by the two wells which precede the plated aprons. The plates involve greater first cost and require more attention than the wells, and it is only in exceptional cases where the ore contains a small proportion of sulphides, and the gold is comparatively free, that wells can compete with plates; but where, as in this instance, they are found to do the work required of them, they are to be preferred for the reasons given.

It is the custom to add a bucketful of lime to each 10 heads every two hours, as this is found to prevent the formation of a black scum on the amalgamating-tables due to the presence of base sulphides in the ore.

At the Britannia United mill, located on Bakery Hill, near the spot where the Welcome nugget (weighing 2,159 ounces and sold for £10,500) was found on June 15th, 1858, the ore treated is more free-milling than that of the Star of the East mine. This is proved by the lower percentage of the concentrates, the greater fineness of the gold, and the increased retort yield. The bullion runs exceptionally high, being worth £4 3s. per ounce, equivalent to 978 fine, and owing to the coarseness of the gold the retort percentage averages 50, while it sometimes reaches 65.

The arrangement of the mill is very similar to that of the new mill of the Star of the East mine. One ounce of mercury is added to the mortar-box per ounce of gold in the ore. Immediately outside the battery there is a well, $1\frac{1}{2}$ inches deep and 3 inches wide, containing 10 lbs. of mercury. The copper plates have a gradient of 1 inch per foot. The blanket-strakes are 16 feet long, divided into three; each strip being 17 inches wide, with

a grade of $1\frac{1}{4}$ inches per foot. Quicklime is added to the battery at the rate of 5 lbs. per 5 heads per 24 hours. The battery water is warmed by conducting the condenser water of the engine into the tank which supplies the mill. This practice of using the condenser water in the battery is decidedly bad, for it is certain to carry grease with it.

At the New Chum Consolidated mill at Bendigo, where condenser water is used in the battery, an examination of the launder which carries it outside the mill showed that the bottom and sides were coated with a slimy ooze which must be prejudicial to amalgamation. At the Britannia United mill, using condenser water, the loss of mercury is only 2·7 dwts. per ton of ore, much less than at the Star of the East mill, where condenser water is not employed. These seemingly contradictory results may be explained by the addition of quicklime, which as an alkali is a solvent of grease, and, though not intended as an antidote for the greasy matter in the condenser water, no doubt acts as such. At the Britannia United mill it is added direct to the ore fed to the battery. At the Star of the East mill it is added only, as before stated, to the blanketings previous to their re-introduction into the coffer. It is in general use at Ballarat for the purpose of neutralizing the acidity of the battery water (produced by the partially decomposed sulphides in the stamper-boxes) and keeps the amalgamating-tables in good order, and prevents corrosion of the screens. The desirability of warming the battery water in a climate like that of Ballarat is, however, open to question.

At the alluvial mines of Otago, New Zealand, the use of mercury is hardly known, the explanation given being that mercury will not act in the cold of that region. This is due to the use of hot water in cleaning-up at both mines and mills. The idea is, of course, erroneous, but there is a substratum of truth in it; for amalgamation is usually assisted by heat, and retarded by cold, but only within narrow limits.

At Black Hawk, Colorado, at an elevation of over 8,000 feet, the millmen say that the bitter cold of winter is better for amalgamation on the copper plates than summer heat, because heat thins the amalgam, and the vibration of the mill, caused by the stamps, tends to make the globules of mercury run off and down the surface of the tables. Cold, on the other hand, thickens the amalgam, and tends to keep it in position; nevertheless, in cases of extreme cold, as in Dakota, the water must be, of course, warmed. Hot water has also one beneficial effect in another way, as slimes which will float in cold water will sink in water which is warmed, owing to the expansion of the air-bubbles, that float the fine dust and create slimes.

On the whole, however, while gold amalgamation is benefited in this way by heat, yet below the temperature of boiling water the effects of a small rise are so slight that it is doubtful if the use of warm water be advisable in ordinary gold stamp-milling. It is certainly not to be recommended in a locality with a summer temperature like that of Ballarat, which is often 75 degs. in the shade, when the water in the mill would in fact be giving off vapour.

The writer believes that in tropical climates an advantage might sometimes be found by cooling the water, say, by storing it in underground tanks.

At the New Normanby mill in East Ballarat the ore treated is of a very free milling nature, the gold which occurs in quartz is almost free from pyrites, and is of a somewhat coarse character, as proved by the retort percentage, the yield of bullion being seldom under 55 per cent., and averaging very nearly 70 per cent.

The North Cornish mill is at Daylesford, in the mining district, but not in the town, of Ballarat. The amalgamation is effected by methods similar to those described as in use at the Star of the East mill. Scarcely 11 per cent. of the amalgam obtained comes from the mortar-boxes, most of it being derived from the skimmings of the wells and the blanket sands. The latter are treated in a set of six berdans, the tailings from which go to a Frue vanner. Up to the end of 1890, on a paid-up capital of only £4,000, the North Cornish property had paid dividends amounting to £85,500, but the mill is stated to be miserably incomplete. Although very favourably situated as regards fall, it is unprovided with rock-breakers or automatic feeders, or with a sufficient number of concentrators (Frue vanners), the ore being one that requires very careful concentration. Fourteen vanners are all that are used, whilst 22 of these machines is the least number that should be employed for good work.

The New Star of the East mill and the Britannia United have exceptionally heavy stamps, following a tendency observable in Californian practice some years ago, which has been abandoned in favour of reversion to a lighter pattern. This is an instance of the needless expenditure of time and money in trying experiments which some other district has already carried out. There is no worse waste than the waste of experience in going over the same ground twice.

In the Britannia United and New Normanby mills the gratings are coarser, and the stamps make fewer drops than the others, owing to the gold being coarser and of a somewhat free character. The unusual coarseness of the gold and the absence of pyrites worth concentration

render rougher stamping permissible, and account for the low drop of the New Normanby as compared with the others.

The North Cornish mill uses the finest gratings, since the gold in the ore it treats is the most associated with pyrites.

In the depth of discharge there is the same wide variation to be remarked as in most colonial mills, a factor the importance of which in milling is too little appreciated all the world over.

In screens the colonial mills have been keeping to the follow-my-leader policy, which is the keynote to all that is deficient in their modes of milling. Actual practical tests show that wire-cloth has a much larger area of discharge than punched sheet-iron. The capacity of many of the Ballarat mills would be increased 10 to 20 per cent. by using wire-cloth gratings, and the small extra cost would be more than compensated by the larger duty of the batteries. The full benefit of wire-cloth screens can be best obtained by having a double set of gratings, so that while one is in use, the other can be dried and cleaned with wire brushes.

The closer work done by the Frue vanners (as compared with ordinary percussion-tables) assists in keeping up the grade of the concentrates at the North Cornish mill, which treats the most refractory ore of any of this group of mills.

The fineness of the bullion tells the same story, that of the Ballarat mills being of unusual purity and finer than the Daylesford bullion.

The very shallow but variable depth of discharge of the North Cornish mortar explains the short life of the screens, which only last, on the average, 6 days.

The top of the dies (when put in new) is flush with the bottom of the screen, and the splash is, in consequence, very violent. The dies are allowed to wear down 6 inches. It is probable that the extraction of the gold would be benefited by leaning towards the greater rather than the lesser depth of discharge, and keeping it uniform.

At the Britannia United mill the average depth of issue is no greater, but it varies between 1 and 2 inches. This, coupled with the fact that the gratings are of coarser mesh, explains why they last more than double as long as at the North Cornish mill.

As the New Normanby and Britannia United ore contains a minimum quantity of sulphides, and the ore is crushed coarser, the consumption of mercury is less than that of the other mills.

The quantity of water used varies with the gradient of the amalgamating-tables, which must be regulated by the heaviness of the pulp ; it is also largely dependent on the extent of blanket surface. The North

Cornish mill uses the least water, since the blankets, being followed by Frue vanners, are shorter than those of the other mills using percussion-tables.

At Ballarat the ample capital and adequate ore-supply remove all excuse for the incompleteness of the mills, so far as concerns appliances and arrangement for the automatic handling of the ore. In the feeding of the 50 stamps of the North Cornish mill, 5 men at 30s. per week are employed per shift, representing a cost of £1,125 per annum. For mining companies like the Star of the East and the North Cornish, both owning magnificent mines, paying large and regular dividends, and possessing very considerable ore-reserves, there can be no excuse for the non-employment of rock-breakers, ore-feeding machines, and a proper and adequate concentrating-plant. They stand as monuments of what should be more truly called obstinate ignorance and perverse disregard of modern experience, than dignified by such a misused word as conservatism. It is greatly to be regretted that for reasons all of them illogical and untenable, the mills of such an important mining district should be so out of date and incomplete. In conclusion, therefore, it must be said that while the actual extraction is excellently carried out, the mills of Ballarat are woefully below the ideal, both in the handling of the ore which immediately precedes stamping, and in the after-treatment which succeeds amalgamation.

Practice in the Ovens District, Victoria.

The Ovens district is north-east of Melbourne, near the border-line dividing Victoria from New South Wales. The mines are scattered over one of the most mountainous parts of Australia, and the mills are located beside streams flowing perennially, in pleasing contrast to the dusty dryness which characterizes most of the mining centres of Queensland, New South Wales, and Victoria. The mills are mostly dependent on custom-work, and are unusually small. The accompanying comparative table, given on the opposite page, indicates their chief features.

The Harrietville mill is the most important and largest in the district, and is the only one that does not do custom-work. It is the property of an English company, who own an extensive group of mines, embracing the Tiddle-Dee-Diddle-Dee, Jackass, Mons Meg, etc. The mortar-box is 5 inches deep, and the dies (which are octagonal) are 4 inches thick. When new the depth of discharge varies from $\frac{1}{2}$ to 1 inch, depending on the amount of sand-packing underneath them. On measuring the depth of discharge in several batteries at the close of a month's work, they were

$2\frac{1}{4}$, 3, 1, $1\frac{1}{4}$, 1, $2\frac{1}{4}$, and $1\frac{3}{4}$ inches, or an average of $1\frac{3}{4}$ inches. The shoes are $9\frac{1}{2}$ inches in diameter and 9 inches long. A shoe weighs 172 lbs., and a die 84 lbs. Both shoes and dies are of best quality white-iron, fagotted, not cast, costing 16s. per cwt. The shoes wear evenly, but the dies irregularly (cupping). Though the ore is comparatively soft, the average wear of iron per ton of ore crushed is 9·8 ounces of the shoe and 3·4 ounces of the die, the wear of the former being excessive. This is to be attributed to the absence of a rock-breaker, and the use of similar material for both shoe and die. The metal of the die should be less hard and more tough than the shoe, an arrangement which would prolong the life of the latter, and promote more even wear of the surface of the die itself; whilst by breaking the mill-stuff to a more uniform size in a stone-breaker, the shoe would be saved much of the violent abrasion that it otherwise undergoes.

	Name of Mill.				
	Harriet-ville.	Orient-al.	Hills-borough	Rail-way.	Ste-phens.
Stamps—					
Number	25	16	8	20	6
Weight (lbs.)	700	784	784	720	840
Number of drops per minute	70	55	60	60	50
Height of drop (inches)	8	9	$9\frac{1}{2}$	9	$9\frac{1}{2}$
Depth of discharge (inches)	2	3	3	4	$4\frac{1}{2}$
Duty per head (tons)...	$1\frac{1}{2}$	$1\frac{1}{3}$	$1\frac{3}{4}$	$1\frac{3}{5}$	$1\frac{1}{2}$
Capacity of mill (tons)	37	20	14	32	9
Description of grating	Rou	nd pu	nched	Russia	iron.
Fineness of grating, in holes per square inch	240	220	200	200	250
Percentage of concentrates	1	†	2	†	†
Gold contents of concentrates (ounces) ...	5	†	$1\frac{1}{2}$	†	†
Fineness of bullion (per 1,000)	965	940	940	950	945
Retort percentage	36	52	50	45	48
Wear of gratings (days)	18	17	18	20	18
Loss of mercury per ton of ore (dwts.) ...	19*	8	4	8	8
Consumption of water per stamp per minute (gallons)	5	4	$3\frac{1}{2}$	4	$4\frac{1}{2}$

For breaking ore in the first instance, a jaw-crusher must necessarily be better adapted to the purpose than a falling weight like a stamp (fine-crushing is not in question), and by compelling the latter to do work which should have been previously done by the former, the material of shoes, dies, screens, and other wearing parts is wasted, the crushing capacity of the mill is diminished, and amalgamation is checked.

The coffer (mortar-box) is of peculiar design, and is provided with an end-discharge. It is 4 feet 6 inches long and 1 foot wide, the interior

* Including pans. † Not saved.

being protected by a cast-iron lining 1 inch thick in four pieces. The bottom of the mortar is flush with the table outside, and it is provided in front with a removable section for cleaning it out.

The grating-frame is in three sections, the two end sections being curved, but each representing a discharge-area corresponding with the one in front, which is 2 feet long. The end gratings have 175 holes per square inch, and the front ones 240. They wear rather longer than those at the end. A set lasts from three weeks to a month, putting through 125 to 200 tons of ore. Complaints in this and other mills are made of the weakening of the gratings along perpendicular lines, caused by the press used in their manufacture. The gratings are not fixed to a frame, but are held in by the latter, which is made of a flat strip of iron bent to the shape of the opening in the mortar-box, and fastened to it by claw-clamps attached to strengthening-ribs, which separate the middle from the side-openings.

The two ends of the mortar-box make a curve, which is an arc, struck from a point in the back of the box. This arrangement of the mortar admits of employing in front, copper tables of unusual width, and by so doing increases the area of amalgamating-surface and the thin distribution of the pulp. An amalgamating table 6 feet wide and 12 feet long is a sight to gladden a millman's heart. The wash of the pulp over the tables is very regular in speed, and even in distribution. The end discharge is generally condemned, and for this reason : that it is ordinarily attempted under the unfavourable conditions of a mortar-box of rectangular shape ; the result being that the issue from the end gratings is weak and irregular, whilst the discharge from the front is injuriously affected. These difficulties are overcome at Harrietville by the variation in shape of the mortar-box, and by using screens of larger mesh at the ends than in the front, as the splash against the former is less than in the front of the box.

The gold-saving is done by collection in the mortar-box, by outside tables and mercury-wells, and indirectly by a modified variety of the Rittinger percussion-table : the concentrates collected being first roasted and then ground in pans. Though there are no amalgamating-plates inside the coffer or mortar, mercury is fed into the battery in variable quantities, depending on the richness of the mill stuff. The average amount is about 1 ounce per battery of 5 stamps every half hour.

The amalgamating-tables have a slope of $\frac{7}{8}$ inch per foot, and are subdivided into four sections lengthways. The first, or apron-plate, is 22 inches long, and the one next below is 3 feet long. These are lined with silver-plated copper plates, whilst those below are usually covered with

plain copper plates. A variation in this arrangement, however, enables one to make an interesting comparison of the effectiveness of plain *versus* silver-coated plates.

In front of No. 5 battery the succession is (1) silver-plated copper plates; (2) plain copper plates; and (3) plain copper plates. The middle plate of this set came from the old Mons Meg mill, and was thoroughly amalgamated and in first-class order. In front of No. 4 battery (next to No. 5) the usual order of two silvered copper plates, followed by two plain copper plates, obtained. After a year's work on ore of uniform character it was found that the plain copper plate forming the third plate of No. 5 battery was well amalgamated, while the corresponding copper plate in front of No. 4 was scarcely whitened by amalgam, proving that the second copper plate of No. 5 had not arrested gold and amalgam as successfully as the corresponding silver-plated copper plate, at No. 4 battery.

Following the first copper plate there is a drop-well, 3½ inches deep, succeeded by a shallow ripple, whilst each of the copper plates below is followed by a ripple of similar kind. All are charged with mercury, and cleaned up once a week. Each battery requires 3 bottles of mercury; or 15 bottles is the stock needed by the entire mill. The loss of mercury is about 75 lbs. avoirdupois per 1,100 tons of ore, or 19 dwts. per ton, including the pan-amalgamation of the roasted concentrates. The consumption in actual milling would be about 8 dwts. per ton.

There are five shaking-tables running at 135 strokes per minute, one to each battery.

The ore yields about 1 per cent. of concentrates, chiefly iron pyrites, and the concentrates run 5 ounces of gold per ton, or at the rate of 1 dwt. per ton of crude ore.

The general clean-up on the last day of each month takes 6 hours; the sand collected in the battery being washed over the amalgamating-tables, and thence passed over the shaking-tables. The heaviest portion is returned to the battery on re-starting.

Thirty-three per cent. of the amalgam obtained is collected from the boxes, and of the remaining two-thirds, 8 per cent. comes from the wells, and 58 per cent. from the amalgamating-tables. Of what is saved by the plates fully 90 per cent. is caught in the first length. Of the total yield of gold 86 per cent. is extracted by direct amalgamation, and 7 to 12 per cent. from the concentrates.

The retort yield of bullion varies from 25 to 52 per cent.

The end discharge and clayey nature of the ore explain the large consumption of water.

The framework of the mill consists of light iron standards, and owing to the care taken in building the foundations, the vibration is not excessive. The tappets are keyed on, as it has been proved that the old-fashioned screw-tappet, though very excellent in theory, and answering admirably when first put in, as soon as it becomes at all loose, (which eventually is certain to happen,) is soon worn out and ruined, necessitating frequent stoppages to cut fresh threads.

The mill, with the exception of being unprovided with a rock-breaker, is an excellent one and excellently managed.

The concentrates are roasted in an ordinary reverberatory furnace, with a hearth 27 feet long and 9 feet wide. It is subdivided in its length into three divisions, with a drop between each of 2 inches. The charge is 12 cwts., taking 8 hours to roast, and the daily capacity of the furnace is 4 tons.

The amalgamation-plant, consisting of two Wheeler pans and a settler, is just below and in front of the roaster, and its extraction is exceedingly good, ranging from 90 to 97 per cent. of the gold in the concentrates, as shown by assay.

It seems probable that if the Harrietville mill were equipped with rock-breakers and self-feeders, and a front-discharge mortar with screens of 175 mesh, it would crush more than $1\frac{1}{2}$ tons per 24 hours, with 700 lbs. stamps given an 8 inches drop and 70 drops per minute, or at any rate with slightly heavier stamps falling more rapidly from a less height.

The extreme thinness of the mortar-bottom and excessive weight of the stamp-shoe (172 lbs.) in a 700 lbs. stamp are to be remarked. In America an 850 lbs. stamp usually has a shoe of 125 lbs. The Australian proportion therefore gives an extremely light stem ($2\frac{1}{2}$ to $2\frac{3}{4}$ inches) or tappet, details in which American practice has been controlled entirely by results of experience.

The reason for using finer screens in front than at the sides of the Harrietville mortar is the desire to equalize the volume of discharge. The intention of the end discharge is to cause an even distribution of the pulp over the amalgamating-tables, whose excessive width is thus rendered serviceable. The suggestion has been made, that a copper plate 6 by 8 feet might be better than one 4 by 12 feet, and it seems very sensible. The copper plate should be as wide as, if not slighly wider than, the screen discharge.

As a rule the mortar-opening exceeds 4 feet. If the amalgamating-table is much wider than the discharge, it will be found hard to obtain an even distribution of pulp. The other extreme is, however, more common. Too often plates are less wide than the discharge. In one mill they were

$2\frac{3}{4}$ inches less than the latter, causing a decided eddy along the sides of the table, owing to the pulp being confined to a narrower space than it was discharged from. The increased force of the current thereby caused was certainly prejudicial to the arresting of the gold. In all milling operations the sooner the gold is caught the better, therefore a table which is wide, but over which the pulp is evenly distributed, and which consequently arrests the gold early in its career over the amalgamating-surface, is to be preferred to one less wide but longer, whose gold-saving is continual for a greater distance from the battery. An amalgamating-table is rarely too long ; it is poor economy to have it even a little too short. An old copper plate can generally be sold for more than its first cost, and therefore any supposed economy in this direction should be disregarded.

The Combination Process.

The plant and process of treatment which obtains at the re-habilitated mill of the Standard Consolidated Mining Company, of Bodie, California, contains many points which are of great interest, as it differs in some respects from the ordinary, and exhibits another phase of Californian practice, affording us an illustration of the combination process.

The Standard mill contains 20 stamps, and appears to have cost £10,800, or £540 per stamp. Sundry additions were made to the original machinery by the erection of two boilers, 16 feet long and 54 inches in diameter, with brickwork chimney. The mill was originally provided with only one set, necessitating a stoppage of the entire mill for 18 to 24 hours every two months for repairs, which entailed a proportionate loss of product, while expenses ran on as usual. The cost of putting in these boilers was £120, whereas a stoppage of the mill for 20 hours involved a direct loss of £80 to £120 ; consequently this provision should more than pay for itself in four months. The plant was also increased in 1891 by fitting up two pans and a settler (for the treatment of concentrates), and the addition to the continuous system of two pans and another pointed settling-box.

The ore is crushed wet, and run over silvered copper plates, which, according to the records of the six months prior to the issue of the company's report on January 31st, 1892, took out 80 per cent. of all the free gold and silver contents. The major part of the remainder stays in the batteries, being too coarse and heavy to be thrown out through the screens, and is extracted by amalgamation in the clean-up pan at the end of every month.

The pulp leaving the plates is run over four Frue vanners with belts 6

feet wide, and the tailings from these are led into pointed boxes in order to get rid of the surplus water from the batteries and vanners before treatment in the pans. The thickened pulp discharged from the settling-boxes is raised by means of a bucket elevator to a series of eight pans and three settlers arranged on the Boss continuous system. From the settlers, after passing through an agitator, the tailings escape into the tailing-pits.

The summary of mill work for the year 1891 shows 15,704 tons of ore crushed. The average crushing during the first five months was 1,249 tons, during which time a No. 10 (corresponding to 40 mesh) Russian iron slot-screen was used on the batteries. The average crushing for the last seven months was 1,351 tons, or a gain of 102 tons per month accomplished by the use of a No. 0 (corresponding to 40 mesh) tin punched screen. These latter cost 2s. 10d. each delivered, and two of them will outlast the average Russian iron screen, which costs 13s. 8d.

While battery samples are taken regularly at the mortar-lip, they serve simply as an indication of the grade of the ore, but do not represent its true value, on account of the heavier particles of gold remaining in the mortar, as already explained.

The total value of the ore is obtained by adding the bullion product of copper plates and batteries, the amount of monthly output of concentrates (calculated from daily assays of weighed amounts) and the value in the vanner tailings, this latter item being the product of the number of tons of ore crushed, multiplied by the average tailings assay for the month.

While the percentage of extraction is not high, it is, according to the few available records, from 4 to 5 per cent. greater than that obtained on this ore previous to the present system of milling, and exceeds that of 1890 by 3 per cent.

Analysis shows the ore to contain no base metals other than a very small amount of oxide of iron; nevertheless, it is only partially free-milling, and the percentage of extraction is consequently low.

Richer ore, such as that milled in the spring and summer months, gives higher results on account of carrying a higher proportion of the precious metals in a free state. While, treating a lower-grade ore, as during the last half-year of 1891, the extraction could only be maintained by the partially-successful amalgamation of tailings in the continuous pans. Working on a lower-grade ore a higher extraction was obtained than in previous years, which is certainly satisfactory considering that the ore-treatment is here limited to a certain line of milling operations by the conditions which obtain of high labour, expensive fuel, and costly freights.

The following table shows the expenses for 1891 :—

Month.	Cost per Ton of Ore.			
	Mining.	Milling.	Superintendence, Bullion Freight, Insurance, Taxes, Office Expenses, etc.	Total.
	Dollars.	Dollars.	Dollars.	Dollars.
1891. February ...	6·208	2·982	0·475	*9·665
March	7·665	3·742	0·366	11·773
April	7·784	3·973	0·334	12·191
May	7·840	3·830	0·524	12·194
June	8·149	4·930	0·854	†13·933
July	6·005	4·361	0·876	11·242
August	7·042	3·388	0·564	10·994
September ...	6·727	3·719	0·527	10·973
October	7·770	4·570	0·610	‡12·950
November ...	6·904	3·903	§1·418	‡12·225
December ...	6·842	4·013	0·554	11·309
1892. January ...	6·295	3·370	0·675	10·340
Averages for the year	7·080 29s. 6d.	3·890 16s. 2½d.	0·650 2s. 8½d.	11·620 48s. 5d.

The accompanying statement, showing the cost per ton of mining, milling, and all incidental expenses, exemplifies the expensive conditions alluded to. That milling is being done at a much reduced cost is shown by the fact that 15,704 tons of ore were crushed in 1891, at a total milling expenditure of 61,002·94 dollars, or 3·89 dollars per ton, while in 1889 only 11,498 tons had been crushed at a cost of 60,000 dollars, or over 5·20 dollars per ton.

AMALGAMATION OF 86 TONS OF CONCENTRATES.

86 Tons.	Assay Value per Ton.			Contents based on Assay of each Pan Charge.			Percentage of Extraction	Loss of Mercury per Ton.
	Gold.	Silver.	Total.	Gold.	Silver.	Total.		
	Dols.	Dols.	Dols.	Dollars.	Dollars.	Dollars.	By Assay. Per Cent.	Lbs.
Concentrates ...	67·53	28·18	95·71	5,807·28	2,423·49	8,230·77	74 $\frac{6}{10}$	2
Tailings ...	15·34	8·77	24·11	1,318·99	754·70	2,073·69	Actual.	
Assay difference	52·18	19·41	71·59	4,488·29	1,668·79	6,157·08	78 $\frac{8}{10}$	
	Actually extracted per ton of ore.							
Actual bullion returns	55·62	19·11	74·73	4,783·56	1,643·39	6,426·95	78 $\frac{6}{10}$	
	Actually contained in tailings per ton of ore.							
	11·91	9·07	20·98					

* Cost exceptionally low, in consequence of not including the entire month's expenses, the bullion product being unusually small.

† Cost unusually high, due to payment of all outstanding bills.

‡ Increased, on account of laying in winter supplies for mine and mill.

§ This item higher due to payment of taxes.

The four concentrators produce 10½ tons of concentrates per month, carrying 80 to 100 dollars per ton, *i.e.*, £18 15s. on the average per ton. This represents a contents of concentratable material in the original ore of ⅔ per cent., showing the ore to be hardly of a concentratable character. Nevertheless, the vanners cost so little to operate while furnishing a high-grade product that it pays to continue running them.

There were on hand in July, 86 tons of concentrates, the product of eight months' work. Previously these concentrates had been shipped to the Selby Silver and Lead Company of San Francisco, and on account of heavy freight and reduction costs had yielded to the Standard Company only £9, or 48 per cent. of their value (£18 15s.), this arrangement, therefore, appears to have cost the company £9 15s. per ton.

It being found that the concentrates contained no sulphides and little or no base metal, two pans were fitted up in the mill, and the concentrates were put through a specially adapted slow treatment, with the results shown in the previous statement, and a return of 31 per cent. more than that obtained by shipping to reduction-works.

The tailings from the pans are carefully banked up separately in a reservoir, and will yield a further proportion of their contents after exposure and oxidation.

A Carter magnetic separator, through which the dried concentrates were being passed in order to eliminate the magnetic oxide of iron (of which they contain 5 to 10 per cent.), was discarded, as pan-amalgamation tests showed 16 per cent. higher extraction on the original concentrates than those which had gone through the separator, though the bullion produced was necessarily baser.

The concentrates are now being regularly treated by pan-amalgamation as the best and cheapest method of realizing them quickly.

Chlorination-tests, after roasting with salt and sulphur, have shown no higher percentage of extraction than that of slow amalgamation; moreover, the loss of gold and silver in roasting is considerable (15 to 20 per cent.), and the cost of wood at 10 dollars per cord makes the process more expensive than amalgamation, for which the ore is fitted.

The mercury on the plates and in the pans takes out all the free gold that will amalgamate, the remainder needs a different treatment, and is worked as tailings. For this purpose the mill originally contained six pans and three settlers fitted up for the Boss continuous system, and one pointed settling-box 11 feet by 7 feet by 7¼ feet in size. One or two trial runs had been made previous to 1891, but the product was increased by mixing in some richer material from the blanket-sluices in the Bulwer Standard mill.

The pan capacity being too limited, two pans were added to the above plant in 1891, and another settling-box put in, to take the overflow from the first. Though there is still a loss of slimes in the overflow of the second box, it is impossible to avoid this and retain the proper consistency of pulp in the pans.

It may be here noted that the mill produces an abnormal quantity of slimes, on account of the large quantity of clay accompanying the quartz in the veins. From trial runs of several weeks each it was ascertained that basing the bullion by the use of salt and bluestone in quantity did not increase the extraction, and the needful small amount of chemicals necessary was determined. The tests also showed that much grinding in the pans was to be avoided. Since then the work on the tailings has steadily improved. The average of several chlorination-tests on the raw tailings show about 8 per cent. higher extraction than the continuous-pan system, and for the months of October, November, and December, 1891, and January, 1892, gave a yield of 35 per cent., at a cost of 3s. 8d. per ton.

Most probably, roasting with salt and sulphur, and chlorination by either the Plattner or barrel process, would give a much higher percentage of extraction, but with wood at 10 dollars per cord, labourers' wages at 3·50 dollars per day, and freight rates of 3 and 4 cents. per lb. (the latter has to be taken into consideration on account of chemicals essential to either process), it would be impossible to figure a profit on tailings averaging 7·50 dollars in gold and silver per ton. Large sample lots of tailings were sent to the Denver Gold and Silver Extraction Company (cyanide process) for trial, and to the Noble M. and M. Company, of New York,* in the hope of obtaining a method giving a higher extraction, without roasting or other great expense. The cyanide method after 12 hours' agitation in a $\frac{1}{2}$ of 1 per cent. solution gave only 46 per cent. extraction, which is just 12 per cent. better than the present cheap pan process, a margin that would be more than swallowed up by the greater cost of the process. On 100 lbs. of concentrates sent for trial also, worth 86 dollars per ton, a test of 12 hours' agitation in a 1 per cent. solution gave 88 per cent. extraction, or 9 per cent. in excess of that obtained by pan treatment, a difference which would not cover the cost of the cyanide.

During the year ending January 31st, 1892, 16,336 tons of ore was mined, including 8,658 tons of filling from the old stopes. The total cost of working per ton was:—Mining, 6·14 dollars; ore transport, 0·113

* A 200 lbs. sample of vanner-tailings when treated, gave only an extraction of 38 per cent.

dollar ; milling, 3·525 dollars ; superintendence, etc., 0·804 dollar ; a total of 10·582 dollars. The mill ran 357½ days, crushing 16,336 tons of ore, the average duty of the stamps being 2·42 tons per day. The stamps weigh 750 lbs., have a drop of 6 to 7 inches, and a normal speed of 104 drops per minute. The average value of the ore was 17·93 dollars in gold and 1·99 in silver, or 19·92 in all. The amount saved was 72·8 per cent. Of the free gold 85·5 per cent. was obtained from the apron-plates, and 14·5 per cent. from the battery-sands. There was 123½ tons of concentrates treated, having an average value of 58·97 dollars in gold, and 30·86 dollars in silver, a total of 89·83 dollars per ton. These concentrates yielded by pan-amalgamation 80·6 per cent. The tailings left to weather from the previous year were ploughed over several times, and then amalgamated, 134 tons being put through.

A very base bullion was purposely made, barely 100 fine in gold and silver, but the extraction was very good, averaging 67 per cent., and raising the total extraction on the original concentrates to 93 per cent. The yield of the mill tailings treated in the Boss continuous pans was 67·2 per cent. of their assay value, or an average of 12·47 dollars per ton. The use of acid, in place of salt and bluestone which have proved efficacious, was discontinued as it was found more expensive, and gave no better results.

(To be continued.)

The meeting then closed.

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,

FEBRUARY 10TH, 1894.

MR. A. L. STEAVENSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings on January 27th, and that day.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. THOMAS ADAMSON, Assistant Manager, E.I.R. Collieries, Bengal, India.

Mr. WALTER BELL, Colliery Manager, 23, Windsor Terrace, Newcastle-upon-Tyne.

Mr. CHARLES BUTTERS, Mining and Metallurgical Engineer, Rand Central Ore Reduction Co., P.O. Box 1891, Johannesburg, Transvaal.

Mr. THOMAS CARR, Colliery Manager, Bengal Iron and Steel Co., Limited, Colliery Manager's Office, Burrakur, East Indian Railway, Bengal, India.

Mr. RICHARD E. CHISM, Mining Engineer, 3A, Independencia No. 1, City of Mexico.

Mr. NAPIER COCHRANE, Colliery Manager, Aldin Grange, Durham.

Mr. G. D. DELPRAT, Mining Engineer, c/o Messrs. Hill, Delprat, Ferdinand, & Carr, Campanas No. 15, Cordoba, Spain.

Mr. FRED. W. HALL, Colliery Manager, Haswell Lodge, Sunderland.

Mr. GEORGE E. HARRIS, Mining Engineer, Margherita, Debrugarh, Upper Assam.

Mr. JOHN HOWARD JOHNSTON, Civil Engineer, Establecimiento Minéral de Casapalca, Lima, Peru.

Mr. JOSEPH COOK NICHOLSON, Manufacturer of Engineers' Tools, 59, Side, Newcastle-upon-Tyne.

Mr. R. PRICE-WILLIAMS, Civil Engineer, Victoria Mansions, 32, Victoria Street, London, S.W.

Mr. JAMES PROUT, Mechanical Engineer, 16, Claremont Road, Redruth, Cornwall.

Mr. FRANCIS BELL FORSYTH RHODES, Superintendent, National Smelting and Refining Co., South Chicago, Illinois, U.S.A.

Mr. Charles Robinson, Mine Owner and Prospector, Steries Cottage, Marias Road, Cape Town, South Africa.

Mr. Alexander Smart, Assistant Engineer, De Beers Mine, Kimberley, South Africa.

Mr. Frank K. Sykes, Colliery Manager, Springwell Villa, Bishop Auckland.

Mr. Thomas Jackson Woodburne, Mining Engineer, Kimberley Mine, Kimberley, South Africa.

Associate Members—

Mr. Archibald Blue, Director, Bureau of Mines, Toronto, Province of Ontario, Canada.

Mr. John Goethe East, Ship Share Broker, 26, Side, Newcastle-upon-Tyne.

Mr. James I'Anson, Fairfield House, Darlington.

Mr. Arthur C. Payne, Royal College of Science, South Kensington, London, S.W.

Associates—

Mr. Michael Heslop, Under Manager, Rough Lea Colliery, Willington.

Mr. Thomas Wallett, Deputy Overman, Heworth Colliery, Felling, R.S.O., Durham.

Student—

Mr. Frederic Lancelot Booth, Mining Student, Ashington Colliery, Morpeth.

The following gentlemen were nominated for election:—

Members—

Mr. William Remsen Appleby. Professor of Mining and Metallurgy, University of Minnesota, 911, Fifth Street, S.E., Minneapolis, Minnesota.

Mr. Richard G. Bell, Engineer, 14, Archbold Terrace, Newcastle-upon-Tyne.

Mr. Nicholas John Bitzos, Mining Engineer, c/o Mr. A. G. Sourlas, Balouk Bazaar, Constantinople.

Mr. Albert Frederick Calvert, Mining Engineer, The Mount, Oseney Crescent, Camden Road, London, N.W.

Mr. H. B. Collins, Mining Engineer, Garscaddin, by New Kilpatrick, Dumbartonshire.

Mr. Takuma Dan, Mining Engineer, Miike Coal Mining Company, Chikugo, Japan.

Mr. Arthur Dickinson, Mining Engineer, Warham Road, South Croydon, Surrey.

Mr. Joseph Dobbs, Colliery Owner, Jarrow Colliery, Castlecomer, County Kilkenny.

Mr. Uriah Dudley, Mining Engineer, Broken Hill Proprietary Company, Broken Hill, New South Wales.

Mr. Henry Cawood Embleton, Mining Engineer, Methley, near Leeds.

Mr. A. Goerz, Mining Engineer, P.O. Box 1961, Johannesburg, Transvaal.

Mr. Francis Hodge, Mining Engineer, 7, Roskear Villas, Camborne, Cornwall.

Mr. Edward Hooper, Mining and Civil Engineer, c/o Mr. J. H. Hooper, College Precincts, Worcester.

Mr. Henry Ryan Lewis, Mining Engineer, 7, Drapers' Gardens, London, E.C.

Mr. Thomas Charlton Renwick, Colliery Manager, Lumley Colliery, Fence Houses.

Mr. Claude Vautin, Metallurgical Chemist, 42, Old Broad Street, London, E.C.

Mr. Alexander Houstonne Ward, Mining Engineer, Coppull, Chorley.

Associate Members—

Mr. Robert Spencer Hawkins, Mining Engineer, 11, Selhurst Park, South Norwood, London, S.E.

Mr. James Hicks, M.S.A., Architect and Surveyor, Redruth, Cornwall.

Associates—

Mr. Harry F. Coux, Assistant Surveyor, Fairfield House, Spennymoor.

Mr. William Crown, Overman, 39, Fifth Row, Ashington Colliery, Northumberland.

Mr. Robert Alfred Harle, Back Overman, Hebburn Colliery, Hebburn, R.S.O.

Mr. Frederick Charles Swallow, Assistant Manager, Wardley Hall, Newcastle-upon-Tyne.

Students—

Mr. Edward Aston Allport, Mining Student, Colliery Office, Medomsley, R.S.O., County Durham.

Mr. Edward Waters, Mining Student, Throckley Colliery, Newcastle-upon-Tyne.

DISCUSSION ON MR. A. H. STOKES' PAPER ON "A SAFETY-LAMP WITH STANDARD ALCOHOL FLAME" ETC.*

Mr. Stokes (H.M. Inspector of Mines) wrote that a number of his lamps were now in daily use and they had so far fully confirmed the statements made in his paper. He had been surprised to find that large air-currents in mines, when examined with the best-trimmed and most carefully-adjusted flame showed no trace of gas-cap; but, when tested with the small alcohol flame they showed clearly defined gas-caps of varying sizes, indicating that such currents were more or less charged with fire-damp. Much had been said with respect to coal-dust in colliery explosions, but when they found air-currents in large mines charged with 2 per cent. of gas, very little dust was wanted to make an inflammable mixture. It was probable that if the return air-currents of mines were tested for low percentages of gas, many managers would be somewhat surprised at the results obtained. It appeared to him, irrespective of what gas tester was

* *Trans. Fed. Inst.*, vol v., page 462.

used, that the testing of return air-currents in mines for low percentages of gas was a subject well worthy of attention and discussion by the members of the Institute.

Mr. M. Walton Brown exhibited, and explained the Stokes lamp, together with the Pieler and Clowes gas-testing lamps. He gave the following comparative table of the indications observed :—

Percentage of Gas.	Height of Gas-cap in Inches. Pieler.*	Clowes.†	Stokes ‡
$\frac{1}{4}$ ...	1·12	0·67	—
$\frac{1}{2}$ ...	2·00	0·71	0·60
$\frac{3}{4}$ ...	2·37	—	—
1 ...	3·12	0·87	1·00
$1\frac{1}{4}$ ...	3·50	—	—
$1\frac{1}{2}$ ...	4·00	—	1·44
$1\frac{3}{4}$ ...	4·37	—	—
2 ...	5·12	1·22	1·68

He suggested that Messrs. Clowes and Stokes should state the heights of the gas-caps for all the percentages recorded in the table. He did not recommend the use of gas-testing lamps, in quantities larger than 2 per cent., as their use was necessarily accompanied with certain risks, which need not be unnecessarily increased. He had recently made at a large colliery in the county of Durham the following comparative experiments on the three lamps as follows :—

Locality.	Lamp.	Height of Flame-cap. Inches.	Gas Present. Per Cent.
Bottom of staple... ...	Clowes	1·1	$1\frac{1}{2}$
„ „	Stokes	1·0	1
„ „	Pieler	3·2	$1\frac{1}{8}$
Old rolleyway	Clowes	1·7	$2\frac{1}{2}$
„ „	Stokes	extinguished by accident.	
„ „	Pieler	5·0	2

He experienced no difficulties in making a test with the Clowes lamp, and relighting the oil flame in less than 30 seconds, and the manager at the mine referred to, was equally expert in its use. The scale fitted in the Clowes lamp was difficult to see, and he suggested that the indications of the Clowes and Stokes lamps would be more readily recognizable if the measuring-scale consisted of opaque lines painted on the lamp at $\frac{1}{2}$ inch intervals, measured from the point at which the tip of the hydrogen or alcohol flames should be adjusted.

Mr. F. Berkley said he had used the Pieler lamp, and endorsed the remarks of Mr. Brown as to the danger of using an alcohol lamp in

* *Trans. N. E. Inst.*, vol. xxxviii., page 179.

† *Trans. Fed. Inst.*, vol. iv., page 444. ‡ *Ibid.*, vol. v., page 467.

more than 2 per cent. of gas, and in mines of high temperature, as the spirit then evaporated freely, and even overflowed, and burned in a dangerous manner on the outside of the lamp.

Mr. W. C. Blackett stated that he had frequently used the Pieler lamp and found it to yield very satisfactory results, and more especially in indicating so little as $\frac{1}{4}$ per cent. of gas. He asked if the Stokes lamp was of a safe form as regarded velocity of current?

Mr. M. Walton Brown said the Stokes lamp was well shielded, and would be found to be safe in currents of high velocity. With regard to the evaporation and overflow of spirit referred to by Mr. Berkley, he stated that the Pieler lamp exhibited was fitted with a sponge, but for which and other precautionary arrangements* the alcohol would overflow when the lamp became heated.

The President said that it was very desirable that one or other of the lamps or the Liveing apparatus should be used frequently in testing the gas contents of the main return airways; but he did not suggest that this examination should be made by the deputies or by the workmen.

Prof. F. Clowes (Nottingham) wrote that, as regarded the remarks made by Mr. Walton Brown, there was no reason why gas should not be tested for above 2 per cent. by the hydrogen lamp. The lamp had been proved to be a perfectly safe one under the most extreme conditions, and could detect and measure accurately percentages of gas varying from $\frac{1}{4}$ to 6. He had been working with a gauge upon the lamp glass, similar to that suggested by Mr. Brown, and had found it satisfactory. He was gratified to learn that Mr. Brown had found the hydrogen lamp easy and simple in use; and also that Mr. Brown had insisted on the necessity of frequent delicate and accurate examinations for gas in the mine. He had elsewhere given the results of tests undertaken to ascertain the relative merits of the three lamps referred to by Mr. Brown. He had employed not only marsh gas, but also fire-damp containing a known proportion of marsh gas. These tests proved that the indications of the lamps were the same in the artificially prepared marsh gas as in the fire-damp containing the same amount of marsh gas. The standard hydrogen flame, after the lamp had burnt for an hour, showed an easily measurable cap with $\frac{1}{4}$ per cent. of marsh gas. The Pieler lamp could not be brought with any certainty to standard adjustment, owing to the hazy and indefinite character of the tip of the flame; in fact something closely resembling the cap produced by $\frac{1}{4}$ per cent. of gas, was always seen over the flame in gas-free air: the caps also suffered severely by the obstruction offered by the

* *Trans. N. E. Inst.*, vol. xxxviii., page 177

gauze; and in the case of the smaller percentages they never even approximately reached the gauge-marks. In one special experiment a cap, actually 4 inches in height, was apparently reduced to 2 inches by the opacity of the gauze. A similar difficulty in setting the flame to standard height was experienced with the Stokes lamp, and for the same reasons. After burning for an hour, no trustworthy cap was seen until $1\frac{1}{2}$ per cent. of gas was present. In view of the uncertainty in setting and maintaining the flame at standard height, it should be remembered that variations in a test-flame imply very much greater variations in the cap indications.

Mr. James Ashworth wrote, regarding the comparative tables prepared by Mr. M. Walton Brown, that some explanation appeared necessary to account for the behaviour of the Clowes and Stokes gas-testing arrangements. Supposing that both testing-flames were of the same height, it could not be denied that the Clowes hydrogen flame would be very much hotter than the Stokes alcohol flame. Thus for the $\frac{1}{2}$ per cent. mixture the cap on the Clowes lamp should exceed that on the Stokes, and accordingly the table shows that the Clowes lamp gives 0·72 inch and the Stokes lamp 0·60 inch. This result was quite in accordance with his experience, and with a rule he had formed on the basis of a long extended number of experiments, viz., that the height and density of gas-caps were proportionate to the heat and dimension of the testing-flame, if the lamps in which they are used were of similar dimensions. Both the lamps referred to are of the Ashworth-Hepplewhite-Gray type. The only material difference being that the one distinguished as the Clowes lamp had a truncated conical glass $3\frac{3}{4}$ inches in height, and the Stokes lamp had a cylindrical glass about $4\frac{1}{4}$ inches in height; otherwise stated the cubical content of the Clowes lamp was very much smaller than that of the Stokes lamp. When the results with 1 per cent. were examined the difference of cubical content appeared to operate and to reverse the rule he (Mr. Ashworth) had stated previously, and the Clowes lamp gave 0·88 inch, and the Stokes 1·00 inch. As the percentage of gas increased so did this divergence of results, and for 2 per cent. the Clowes lamp showed 1·24 inches, and the Stokes lamp 1·68 inches. But the pit tests did not appear to support the correctness of the measured gas tests, for the Clowes lamp gave a 1·10 cap against the Stokes lamp with a 1·00 inch cap, and declared $1\frac{1}{2}$ per cent. of gas present, whereas the Stokes lamp recorded only 1 per cent. He therefore concluded that the cooling effect of the air-current on the large area of the Stokes lamp was most markedly shown, that the vigour of the air-current within the Stokes lamp was much less than within the Clowes lamp, and that the

volume of gas per minute passing over the Stokes flame was therefore much smaller than that passing over the Clowes flame. He (Mr. Ashworth) considered that if a lamp for testing is to give correct and uniform results it must possess a sufficiently large and powerful flame to maintain a vigorous current through the lamp, and if it does not do so the current becomes sluggish. In some cases the cooling effect is so excessive as to make it *nil*, in which case the products of combustion gyrate above the flame, and destroy the cap. He considered the results obtained in his own practical experience so convincing and decisive that he had, in his own arrangement of the Ashworth-Hepplewhite-Gray lamp, adopted a much smaller truncated conical glass, viz., $2\frac{3}{4}$ inches in height for both the combined oil-and-spirit lamp, and the benzolene special gas-testing lamp. Moreover, he considered that, since the use of a lamp with a non-luminous flame, such as that from alcohol, was for a special purpose, and not for general and everyday use, it was better, and in every way more satisfactory, to have only one class of flame in one lamp. It might be remembered that he (Mr. Ashworth) exhibited at Sheffield in January, 1890, a lamp with two burners, one (oil) for illumination and one (alcohol) for gas-testing. In this lamp the alcohol-vessel was placed directly below the oil-vessel, and consequently when the lamp and the oil-vessel became hot the spirit forced its way under the cap which covered the spirit-burner when not in use, and ignited in a similar way to that described by Mr. F. Berkley. He then reconstructed the lamp without a cover over the spirit, and also left an air space between the oil and spirit reservoirs (the latter being the lowest) and then found no difficulty in using the lamp continuously for several hours—the flame being 0·5 inch in height, on a wick of, say, 0·4 inch wide. He found that the flame remained constant in height after 3 hours' use, just as it was when he started. He then used methylated spirit, and was now making a lamp with an equally large spirit flame (but without the oil-reservoir) because a flame of that size keeps up a full current of air through the lamp and practically consumes all the air which the four inlet-tubes of the ordinary Ashworth-Hepplewhite-Gray lamp will supply, and mainly because the testing-flame ought to be kept to one standard height for the whole pit, such standard being the height at the pit-bottom. It will be quite evident after a moment's consideration that no flame, not even hydrogen, can be set to any standard height in a return airway. The testing-flame must be set in the fresh air to obtain a correct comparison, and the testing-flame must be sufficiently powerful to maintain a full current through the lamp without the assistance of an oil flame. With a lamp such as that

above described, measurements just as accurate, if not more accurate, can be made as those with the Pieler lamp, because the cap can be viewed on a shorter scale through the improved glass constructed with a dark non-reflective side, which brings out the cap very distinctly. The Clowes, Stokes, and Ashworth lamps are all fitted to different patterns of the Ashworth-Hepplewhite-Gray lamp, the main difference being in the size of the glass cylinder, and all three types are exceedingly safe lamps and are immediately extinguished in an explosive current. He was surprised to find that a large number of Pieler lamps, which are known to be unsafe in any explosive current, are in use belowground, when safer and more accurate lamps are available. In conclusion, he very strongly recommended that every testing-lamp should have the testing-flame hid from the eye by a fixed screen or shield, as in the Pieler lamp.

Prof. F. Clowes demurred altogether to Mr. Ashworth's statement that the hydrogen flame could not be set to standard height in a return airway. Both in the test chamber and below ground the hydrogen flame was always set to standard height in the presence of the gas, and with unfailing certainty. In fact all the cap measurements had been proposely made under these conditions. He also considered that screening the hydrogen flame during the observation of the cap, as recommended by Mr. Ashworth, was distinctly disadvantageous. The hydrogen flame differed from all other testing flames in being so pale as not to interfere in any way with the perception of the cap, and practical men who used the lamp preferred to see the whole height of the hydrogen flame, as it assisted them in following up the position of the cap. It appeared that Mr. Ashworth wished to persuade them that the carriage of two separate lamps, one for lighting and another for gas-testing, was preferable to carrying one lamp only, which served for the greater part of the testing, and could be rapidly adopted when necessary to make accurate and delicate tests. He hoped that delicate and accurate gas-testing would become a frequent matter and not be the occasional process suggested by Mr. Ashworth.

The discussion was then closed.

Dr. P. P. Bedson read the following paper on "A Contribution to our Knowledge of Coal-dust":—

A CONTRIBUTION TO OUR KNOWLEDGE OF COAL-DUST.

Part II.

By P. PHILLIPS BEDSON, D.Sc. (Lond.), B.Sc. (Vict.), F.I.C., F.C.S.
Durham College of Science, Newcastle-upon-Tyne.

In a paper read before this Institute* the author gave an account of the results of some experiments made with samples of coal-dust collected on the Ryhope screens. These experiments were made with the object of determining whether coal-dust contained "enclosed gases," and to ascertain the nature of these gases. In the paper referred to, the hope was expressed that at some future time the author would be in a position to place before the members of this Institute the results of a further investigation of this question, but he regrets that circumstances beyond his control have so long delayed the publication of the results.

Coal-dust from Ryhope Colliery.

Since the publication of the paper referred to, the examination of the Ryhope coal-dust has been repeated, and in the analysis of the gases more exact and refined methods have been employed than was possible before, by reason of the limited accommodation afforded in the rooms temporarily occupied by the Durham College of Science at that time. But whilst the acquirement by the college of buildings specially well suited both for teaching and for research has enabled the author to continue the investigation under more favourable conditions, still the time occupied in furnishing and equipping the chemical department has contributed in no small degree to delay the completion of the work.

Since in the first paper the manner of extracting the gases from the coal-dust and the methods of analysing such a mixture are described, it will suffice to state that the gases are obtained from weighed quantities of coal-dust, by heating at 100 degs. C. (212 degs. Fahr.) *in vacuo* for several hours ; the volume of gas so produced is next measured, and then its composition ascertained.

In repeating the examination of the Ryhope coal-dust the author has attempted to obtain an insight into the nature of the combustible gases enclosed in this dust, by extracting the gases from the dust when heated

* *Trans. N. E. Inst.*, vol. xxxvii., page 245.

in vacuo at different temperatures, ranging from 30 degs. to 100 degs. C. For this purpose some 670 grammes of the dust were placed in a bottle and attached to the mercurial air-pump. After removing the air, the vessel containing the coal-dust was heated by means of a water-bath for some twenty days at 30 degs. C. (86 degs. Fahr.); the gas produced was drawn off and its volume determined. The temperature was then raised to 50 degs. C. (122 degs. Fahr.), at which it was maintained for some ten days; in this way a second quantity of gas was obtained. A third extraction was made by heating at 60 degs. C. (140 degs. Fahr.) for another ten days, a fourth by heating at 60-80 degs. C. (140-176 degs. Fahr.), for seven days, and, finally, a fifth by heating at 100 degs. C. (212 degs. Fahr.) for some nine hours.

In this manner the following volumes of gas were obtained:—

TABLE I.

	Cubic centimetres at 0 deg. C. and 760 millimetres (32 degs. Fahr. and 29·92 inches.)
First extraction, at 30° C. (86° Fahr.)	100·9
Second ,, ,, 50° C. (122° Fahr.)	160·6
Third ,, ,, 60° C. (140° Fahr.)	116·3
Fourth ,, ,, 60-80° C. (140-176° Fahr.)	286·0
Fifth ,, ,, 100° C. (212° Fahr.)	89·5
Total volume	753·3

The total volume of gas obtained from 100 grammes of coal-dust is, therefore, 112·4 cubic centimetres measured at 0 degs. C. and under a pressure of 760 millimetres of mercury.

The composition of these several samples is given in the following table:—

TABLE II.

	I.	II.	III.	IV.	V.
Carbon dioxide	5·77	8·34	12·12	27·35	20·80
Oxygen	9·33	7·31	5·35	0·56	4·16
Carbon monoxide ...	—	—	—	1·68	2·34
Olefines ($C_n H_{2n}$) ...	—	0·39	0·77	2·14	4·74
Paraffins ($C_n H^{2n+2}$) ...	3·16	4·95	9·39	31·86	29·80
Nitrogen	81·60	79·01	72·37	35·70	38·16
	99·86	100·00	100·00	99·29	100·00
Values of n in the formula ($C_n H_{2n+2}$)	2·1	2 2	2·1	2·3	2·8

In the first place it is to be noted that the combustible gas is not simply marsh gas, for if this were the case, then the value for n would be one, whereas in each case, analysis proved the gas to contain a larger proportion of carbon, or in other words, that we are dealing here with mixtures of members of the series of hydrocarbons to which marsh gas belongs, and

whilst we are not in a position to state definitely what members are present, there is, in the increasing value of n, evidence of the existence in the mixture of marsh gas (CH_4), ethane (C_2H_6), and possibly propane (C_3H_8).

Again, comparing the composition of the first four extractions, which were obtained without dismounting the apparatus, it is seen that the proportion of carbon dioxide increases, and that of the oxygen decreases with the rise in temperature. Further, that whilst some combustible gas was obtained at 30 degs. C. (86 degs. Fahr.), this was comparatively small in amount and consisted entirely of paraffins; the olefines were present only in small amount, and were not expelled until the temperature, at which the coal was heated, rose above 50 degs. (122 degs. Fahr.)

The gas obtained in the fifth extraction cannot, unfortunately, be compared with the others, as it was doubtless contaminated by air, since during this operation a breakage in the apparatus necessitated remounting it. The results, however, suffice to confirm previous experiments with this dust, showing it to contain "enclosed gases;" among which we find hydrocarbons belonging both to the paraffin and olefine series, and the former is represented not by marsh gas alone, but also by higher members of the series.

Imagining these several fractions, obtained at different temperatures, to be mixed together we should obtain a gas which would have the following composition:—

TABLE III.

Carbon dioxide	17·29
Combustible gases	21·17
Nitrogen and oxygen...	61·54
	100·00

Assuming the oxygen present to represent air, we may interpret these results, as follows:—

TABLE IV.

Carbon dioxide	17·29
Combustible gases	21·17
Nitrogen	41·43
Air	20·11
	100·00

COAL-DUST FROM BRANCEPETH COLLIERY.

In April, 1889, an accident occurred at Brancepeth colliery, caused by the ignition of coal-dust in a coal-box or hopper. A few days after the accident the author visited the scene and secured two samples of the coal-dust from the hopper.

Sample No. 1 represented the coal-dust which had accumulated on the wooden beams in the upper part of the box. The dust was examined for "enclosed gases" in the same manner as that from Ryhope, and gave the following results :—

TABLE V.

			Cubic Centimetres at 0° C. and 760 millimetres (32° Fahr. and 29·92 inches).
First extraction, at ordinary temperature.			Volume not measured.
Second „	30° C. (86° Fahr.)	for 11 days ...	137·6
Third „	40–50° C. (102–122° Fahr.)	„ 14 „ ...	315·4
Fourth „	48–52° C. (118–126° Fahr.)	„ 2 „ ...	211·4
Fifth „	47° C. (116° Fahr.)	„ 2 „ ...	176·0
Sixth „	60° C. (140° Fahr.)	„ 6 „ ...	359·3
Seventh „	100° C. (212° Fahr.)	„ 16 hours ...	397·8
Total volume			1,597·5

The weight of coal-dust employed being 713·1 grammes, this volume represents 224 cubic centimetres of gas per 100 grammes of coal.

The following table contains the results of the analyses of these various samples :—

TABLE VI.

	I.	II.	III.	IV.	V.	VI.	VII.
Carbon dioxide ...	0·98	1·53	2·08	3·81	4·6	3·97	13·65
Oxygen	18·63	14·14	13·64	3·60	9·1	8·70	0·64
Nitrogen	80·39	83·92	84·28	92·60	86·3	87·33	85·71
	100·00	99·59	100·00	100·01	100·0	100·00	100·00

In no case was any combustible gas found. The average composition of the gas obtained from this dust was :—

TABLE VII.

Carbon dioxide	5·86
Oxygen	7·51
Nitrogen	86·63
	100·00

Sample No. 2 was taken from the coal-dust which had accumulated in the lower part of the box. The examination of this dust gave the following results :—

TABLE VIII.

			Cubic Centimetres at 0° C. and 760 millimetres (32° Fahr. and 29·92 inches.)
First extraction at ordinary temperature		for 3 days ...	229·10
Second „	30° C. (86° Fahr.)	„ 12 „ ...	80·85
Third „	50° C. (122° Fahr.)	„ 17 „ ...	75·90
Fourth „	60–70° C. (140–158° Fahr.)	„ 6 „ ...	91·00
Fifth „	100° C. (212° Fahr.)	„ 13 hours ...	71·69
Total volume			548·54

The coal-dust used weighed 777·48 grammes, consequently this volume represents 70·5 cubic centimetres of gas per 100 grammes of coal.

The several fractions of gas had the following composition :—

TABLE IX.

	I.	II.	III.	IV.	V.
Carbon dioxide	0·64	1·62	12·53	28·16	53·66
Oxygen	19·16	13·25	0·09	1·17	0·26
Nitrogen	80·20	85·13	87·38	68·81	42·54
Combustible gas	—	—	traces	1·70	3·53
	100·00	100·00	100·00	99·84	99·99

The average composition of the gas from the dust was, therefore, as follows :—

TABLE X.

Carbon dioxide	13·61
Oxygen	10·23
Nitrogen	75·40
Combustible gas	0·76
	100·00

These samples of dust are seen to differ from one another in the volume of gas which they yield, under the treatment to which they were submitted : No. 2 giving only about one-third the volume of gas yielded by an equal weight of No. 1. The difference in composition of the gases obtained from the two samples is not very great, No. 2 containing some combustible gas, in amount so small as to preclude the possibility of speaking decidedly as to its nature, whilst No. 1 yielded no combustible gas.

The difference between the gases obtained from these dusts and those obtained from the Ryhope coal-dust is so apparent as to require no further comment.

A CONTRIBUTION TO OUR KNOWLEDGE OF COAL-DUST.

Part III.

By P. PHILLIPS BEDSON and W. McCONNELL, Junr.,
Durham College of Science, Newcastle-upon-Tyne.

The investigations of Prof. E. von Meyer (*Journ. Prakt. Chem.* [2], v., 146-182 and 407-416; also *Journ. Chem. Soc.*, x., 798), and those of Mr. W. J. Thomas (*Journ. Chem. Soc.*, 1875 and 1876), have shown that certain varieties of weathered coals, cannel and jet hold enclosed gases, which, like those obtained from the Ryhope dust, contain some olefine hydrocarbons and higher or denser members of the series of hydrocarbons to which marsh gas belongs. It therefore appeared desirable to ascertain by direct experiment the nature of the gases enclosed in the coal from which this dust is produced, and also to learn something of the conditions under which these gases exist in the coal.

Ryhope Colliery.

For this purpose samples of coal from the hutton seam, North pit, Ryhope colliery, were collected by one of the authors at a working-place in the pit, at a point some 500 feet distant from that portion of the seam, yielding the coal from which the dust used in the experiments described in the previous papers was obtained. The coal, as hewn, was broken into small pieces, brushed from adhering dust, and packed into previously weighed flasks. The flasks with their contents, after re-weighing to ascertain the weight of coal, were, as required, sealed on to an air-pump of the Geissler type. The apparatus used in these and all subsequent experiments was made entirely of glass, the several parts being fused together to prevent leakage.

Gases extracted from Ryhope Coal.

The weight of coal taken was 220 grammes, which, after the air had been completely removed from the flask, was heated at the temperatures specified in Table I. The gases evolved at the several temperatures were collected, measured, and analysed. The results of the analyses are given in Table II.

TABLE I.

Samples.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Gas given off, reduced to Volume at 0 C. and 760 millimetres of Mercury Pressure (32 Fahr. and 29·92 inches).
I. ...	5	30°–70° C. (86°–158° Fahr.)	247 cubic centimetres.
II. ...	19	70°–92° C. (158°–196° Fahr.)	336 do.
III. ...	44	100° C. (212° Fahr.)	550 do.
IV. ...	24	100° C. (212° Fahr.)	374 do.
V. ...	27	100° C. (212° Fahr.)	293 do.
Total...	119	.....	1,800 cubic centimetres.

TABLE II.

Samples	I.	II.	III.	IV.	V.
	Volumes.	Volumes.	Volumes.	Volumes.	Volumes.
Carbon dioxide (CO_2) ...	0·5	1·2	0·78	0·3	0·8
Oxygen (O_2)	17·5	9·9	7·20	8·0	8·0
Carbon monoxide (CO) ...	0·0	0·0	0·00	0·0	0·0
Olefines (C_nH_{2n})	0·0	0·4	0·00	0·0	0·0
Paraffins (C_nH_{2n+2})	4·8	26·6	22·47	14·5	6·8
Nitrogen (N_2)	77·2	61·9	70·00	77·2	84·7
	100·0	100·0	100·45	100·0	100·3

The analytical results show the combustible gases present in No. I. to be a mixture of hydrogen and marsh gas, in Nos. II., III., and IV. to be marsh gas, whilst the value of n in the formula C_nH_{2n+2}, found for the combustible gas in No. V., indicates the association with the marsh gas of some higher member of this series, in all probability ethane (C_2H_6).

From the above it will be seen that 100 grammes of this coal would give a total volume of 818·1 cubic centimetres of gas, which contained 16·91 per cent. of combustible gas, consisting of a mixture of hydrogen, marsh gas, and ethane. The coal in the course of the experiments lost 4·5 grammes in weight, the greater portion of which loss was due to the water given off on heating.

Gases Extracted from Ryhope Coal after Crushing.

The coal, after it had ceased to give off gas under the conditions already described, was crushed to a fine powder, and again submitted to a further exhaustion. Of the coal crushed in this manner 208 grammes were taken and heated *in vacuo* at 100 degs. C. (212 degs. Fahr.) for 26 hours; at the end of which time a volume of gas measuring 109 cubic centimetres at 0 degs. C. (32 degs. Fahr.), and a pressure of 760 milli-

metres (29·92 inches) of mercury, was drawn off, the composition of which proved to be as follows :—

TABLE III.

	Volumes.
Carbon dioxide (CO_2)	0·85
Oxygen (O_2)	6·95
Carbon monoxide (CO)	trace
Olefines (C_nH_{2n})	1·10
Paraffins (C_nH_{2n+2})	18·40
Nitrogen (N_2)	72·70
	100·00

This additional volume of gas represents 52·4 cubic centimetres for every 100 grammes of coal. The explosion of the combustible gases with measured volumes of oxygen shows that they contain higher members of the paraffin series, and consist, in all probability, of a mixture of marsh gas (CH_4), ethane (C_2H_6), and propane (C_3H_8).

The results of these experiments support the view already expressed by one of the authors, to explain the presence of the denser hydrocarbons among the gases enclosed in the dust produced in the screening of this coal. For it is evident that the enclosed gases contained in this coal are retained by it with varying degrees of firmness. The lighter gases, such as marsh gas and hydrogen are the more readily removed, and the exposure of the fine particles of dust to the air would allow of their ready diffusion into the air, leaving the denser gases held by the particles of coal substance: hence the latter form the chief combustible constituents of the gases enclosed in the dust formed during the screening of the coal.

Gases obtained from Ryhope Coal when Heated at Temperatures above 100 degs. C. (212 degs. Fahr.).

In the light of the results just described it appeared desirable to examine the gases obtained by heating the coal under reduced pressure at different intervals of temperature. For this purpose a weighed quantity of coal was introduced into a glass cylindrical tube ($1\frac{1}{8}$ inches in diameter), sealed off at one end, the other being narrowed down and sealed on to the pump. This tube was surrounded by a second, thus forming a jacket through which steam and other vapours were allowed to pass, for the purpose of raising the coal to the desired temperature. Ninety grammes of coal were used, and submitted to the following treatment :—

A. The coal (90 grammes), after removal of the air, was heated by a current of steam passed through the outer tube for several hours, with the result that the following volumes of gas were produced :—

TABLE IV.

Samples.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Gas evolved, reduced to Volume at 0° C. and 760 millimetres of Mercury Pressure (212° Fahr. and 29·92 inches).
A ...	44	100° C. (212° Fahr.)	80 cubic centimetres.
B ...	34	100° C. (212° Fahr.)	30 ,,
Total ...	78	...	110 cubic centimetres.

Thus 100 grammes of coal is seen to yield at 100 degs. C. (212 degs. Fahr.) 122·2 cubic centimetres gas, and the analyses of the two samples gave the following results:—

TABLE V.

	Sample A.	Sample B.	Calculated Composition of the Total.
Carbon dioxide (CO_2) ...	3·0	4·5	3·4
Oxygen (O_2)	1·0	1·5	1·1
Carbon monoxide (CO) ...	0·0	0·0	0·0
Olefines (C_nH_{2n})	0·0	0·0	0·0
Paraffins (C_nH_{2n+2}) ...	43·4	67·4	49·9
Nitrogen (N_2)	52·6	26·6	45·5
	100·0	100·0	99·9

The analytical results show the combustible gas to contain in addition to marsh gas, probably also ethane, the value for n found being: in sample A, 1·9; in sample B, 1·8.

After the removal of the gas, the heating of the coal was continued for 12 hours, and since the mercury in the pump remained stationary it was concluded that all gas obtainable at 100 degs. C. (212 degs. Fahr.) had been removed.

B. The vapour from boiling amyl-alcohol was next passed through the jacket, and in this manner the coal was heated at 130 degs. C. (266 degs. Fahr.) (the boiling point of the alcohol used) for 14 hours, a further volume of 20 cubic centimetres of gas was obtained, representing 22·2 cubic centimetres if calculated on 100 grammes of coal. The composition of the gas was found to be as follows:—

TABLE VI.

	Volumes.
Carbon dioxide (CO_2)	8·5
Oxygen (O_2)	4·0
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	1·8
Paraffins (C_nH_{2n+2})	9·9
Nitrogen (N_2)	75·8
	100·0

Whilst the proportion of combustible gas is smaller than in the gas obtained at 100 degs. C. (212 degs. Fahr.), there are, however, olefines

mixed with the paraffins, and the latter contain a larger proportion of higher members of the series, the value for n in the formula C_nH_{2n+2}, being 2·37.

C. A further heating of the coal for six hours at 130 degs. C. (266 degs. Fahr.) showed that all gas obtainable under these conditions had been expelled, the coal was next raised to a higher temperature by passing the vapour from boiling aniline through the jacket. In this way the coal was heated at 184 degs. C. (363·2 degs. Fahr.) for some ten hours, and a volume of gas measuring 7 cubic centimetres was obtained, which calculated on 100 grammes of coal represents 7·77 cubic centimetres. The composition of this gas was found to be :—

TABLE VII.

	Volumes.
Carbon dioxide (CO_2)	5·8
Oxygen (O_2)	27·0
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	4·6
Nitrogen (N_2)	62·6
	100·0

It will be noted that there is a decrease in the proportion of combustible gas, and an absence of members of the olefine series. With the small proportion of combustible gas it is not possible to speak with absolute certainty of its nature, still the results obtained by burning this gas with measured volumes of oxygen indicate the presence of that member of the paraffin series of the formula C_4H_{10}, viz., butane. The value for n found being 4·35.

D. A further heating of the coal for six hours at 184 degs. C. (363·2 degs. Fahr.) proved that no more gas was evolved at this temperature, the heating was therefore discontinued, and the coal was then crushed to a fine powder. The powdered coal was heated *in vacuo* at 100 degs. C. (212 degs. Fahr.) for eight hours, and thus 34 cubic centimetres of gas were obtained, representing a volume of 37·7 cubic centimetres calculated for 100 grammes of coal. This gas was found to have the following composition :—

TABLE VIII.

	Volumes.
Carbon dioxide (CO_2)	4·4
Oxygen (O_2)	1·2
Carbon monoxide (CO)	1·0
Olefines (C_nH_{2n})	2·6
Paraffins (C_nH_{2n+2})	37·7
Nitrogen (N_2)	53·1
	100·0

The gas contains not only carbon monoxide and olefines, but also a considerable proportion of paraffin hydrocarbons, the analyses of which

give a higher value for n than that found in any previous case, viz., 5·1, which corresponds to a value such as would be given by pentane (C_5H_{12}).

The above results demonstrate the existence of several members of the paraffin series of hydrocarbons among the gases occluded by this coal, and further, the possibility of separating these to a certain extent by extractions performed at different temperatures, and by altering the physical condition of the coal. The experimental evidence supports the view that in all probability there are present in this coal some hydrogen, marsh gas, and the other members of the series of hydrocarbons as high as pentane.

The above results may be stated in the following manner:—That 100 grammes of coal heated *in vacuo* give the following volumes of the several gases, expressed in cubic centimetres, and measured at 0 degs. C. and 760 millimetres (32 degs. Fahr. and 29·92 inches):—

TABLE IX.

Samples.	100° C. *in vacuo* (212° Fahr.).	130° C. *in vacuo* (266° Fahr.).	184° C. *in vacuo* (363° Fahr.).	Crushed and then heated at 100° C. (212° Fahr.) *in vacuo*.
Carbon dioxide	4·20	1·87	0·45	1·66
Oxygen	1·40	0·89	2·10	0·45
Carbon monoxide	0·00	0·00	0·00	0·38
Olefines	0·00	0·40	0·00	0·98
Paraffins	61·07	2·22	0·36	14·25
Nitrogen	55·53	16·82	4·86	20·05
	122·20	22·20	7·77	37·77

These numbers show at once that not only is the greater portion of the occluded gases given off at the temperature of boiling water, but that this is more emphatically true of the combustible gases. Further, it is evident that mechanical subdivision of the coal favours the release of these gases to a very marked extent. Comparing the gas obtained from the crushed coal with that obtained from the coal-dust collected on the screens, the most notable differences are to be found, firstly, in the total volume of gas, which can be readily understood, inasmuch as a considerable amount of the gas has been extracted already from the coal. Secondly, in the larger proportion of carbon dioxide in the gases from the dust, which seems to indicate that the air absorbed by the dust oxidizes some portion of the coal material, thus forming carbon dioxide, which is retained by the coal. Thirdly, the combustible gases contained in the dust produced from coal which had been previously heated at 180 degs. C. (356 degs.

Fahr.), consist of carbon monoxide, olefines, and paraffins, the latter almost entirely made up of pentane, whilst the paraffin hydrocarbons in the dust produced from coal which had been heated at 100 degs. C. (212 degs. Fahr.) *in vacuo* resemble in composition those found in the dust collected on the screens. There is one other point which calls for remark, and that is the different volume of gas obtained from these two samples of Ryhope coal, which is explained when it is remembered that the first was collected at the working-face as described, and the second was sent from the colliery packed in a wooden box after being brought to the surface in the usual manner. The necessarily prolonged exposure to air would allow for a liberation of some of the occluded gases of the coal.

This somewhat lengthy series of operations upon this variety of coal, show that the greater proportion of the enclosed gases can be extracted from the coal by heating it *in vacuo* at 100 degs. C. (212 degs. Fahr.), and then submitting it after crushing to a further heating under these same conditions. Consequently, in examining other coals, this method of treatment has been adopted.

Two other samples of the coal of the hutton seam have been examined, viz., a sample from this seam at the Hebburn colliery, and a second from another colliery in the county of Durham.

Hebburn Colliery.

The considerable volume of gas and the frequent blowers met with in working the hutton seam at this colliery lend a special interest to the examination of this coal.

Gases Extracted from Hebburn Coal.

The results obtained by heating the coal *in vacuo* at 100 degs. C. (212 degs. Fahr.) are given in the following table :—

Table X.

Weight of Coal.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Water.	Volume of Gas reduced to Volume at 0° C. and 760 Millimetres Mercury Pressure (32° Fahr. and 29·92 Inches).
Grammes.			Cubic Centimetres.	Cubic Centimetres.
256	12	100° C. (212° Fahr.)	10	408·2
	33½	100° C. (212° Fahr.)	2	329·0
	45½		12	737·2
193	19	100° C. (212° Fahr.)	8	396·2
	27½	100° C. (212° Fahr.)	—	101·0
	46½		8	497·2

From this it will be seen that 100 grammes of coal gave 117 cubic centimetres of gas having the following percentage composition :—

TABLE XI.

	Volumes.
Carbon dioxide (CO_2)	1·5
Oxygen (O_2)	1·1
Paraffins (C_nH_{2n+2})	78·0
Nitrogen (N_2)	19·3
	99·9

The analytical results give values for n in the formula of the paraffins, varying from 1·49 to 1·6, which we may interpret as indicating the gas to be a mixture of marsh gas and ethane.

Gases Extracted from Hebburn Coal after Crushing.

The coal, after it had ceased to give off gas, under the conditions described, was crushed to a fine powder, and 190 grammes of this powdered coal were heated *in vacuo* at 100 degs. C. (212 degs. Fahr.) for 33·5 hours, and in this way 113 cubic centimetres of gas, measured at 0 degs. C. and 760 millimetres (32 degs. Fahr. and 29·92 inches) being produced. This volume represents 60 cubic centimetres for every 100 grammes of coal-dust. The composition of the gas is given below: the hydrocarbons present are similar to those found in the Ryhope coal, as seen by the value found for n, viz., 2·38 :—

TABLE XII.

	Volumes.
Carbon dioxide (CO_2)	3·2
Oxygen (O_2)	3·0
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	42·1
Nitrogen (N_2)	51·7
	100·0

Gases Extracted from Hebburn Coal-dust.

The dust collected at the screens at this colliery has also been examined, with the results summarized below :—

TABLE XIII.

Weight of Coal.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Water.	Volume of Gas reduced to Volume at 0° C. and 760 Millimetres of Mercury Pressure (32° Fahr. and 29·92 Inches).
Grammes.			Cubic Centimtrs.	
152	47	100° C. (212° Fahr.)	7	50 cubic centimetres.
133	40½	100° C. (212° Fahr.)	5	40 ,,

Thus 100 grammes of coal-dust yielded 31·6 cubic centimetres of gas, the composition of which proved to be the following:—

TABLE XIV.

	Volumes.
Carbon dioxide (CO_2)	13·05
Oxygen (O_2)	0·86
Carbon monoxide (CO)	0·00
Olefines (C_nH_{2n})	0·00
Paraffins (C_nH_{2n+2})	35·65
Nitrogen (N_2)	50·44
	100·00

The combustion of the paraffin hydrocarbons give 3·14 as the value for n, showing that gas to consist of a mixture similar to that obtained from the Ryhope dust.

A COLLIERY IN THE COUNTY OF DURHAM.

The third sample of coal of the hutton seam was obtained from a colliery which some years ago was the scene of a disastrous explosion, in which coal-dust was supposed to have played an important part.

The following table contains the results of the extraction of the gases from this coal at 100 degs. C. (212 degs. Fahr.) and *in vacuo*:—

TABLE XV.

Weight of Coal taken.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Gas given off, reduced to Volume at 0° C. and 760 Millimetres of Mercury Pressure (32° Fahr. and 29·92 Inches).
Grammes.			
76	48·5	100° C. (212° Fahr.)	132·0 cubic centimetres.
194	27·0	100° C. (212° Fahr.)	302·6 ,,
270	75·5		434·6

From this it will be seen that 100 grammes of coal gave 160·9 cubic centimetres of gas, having the following percentage composition:—

TABLE XVI.

	Volumes.
Carbon dioxide (CO_2)	2·0
Oxygen (O_2)	0·9
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	87·5
Nitrogen (N_2)	9·6
	100·0

The combustion of the paraffin hydrocarbons gave from 1·04 to 1·16 as the value for n, showing the combustible gas to be marsh gas.

Gases Extracted from the Coal after Crushing.

The coal, after it had ceased to give any gas under the conditions already described, was crushed to a fine powder, and 208 grammes of this powdered coal was placed in a flask from which air was completely removed. In this case it was noticed that the crushed coal gave off some gas at the ordinary temperature. The volume so obtained proved to measure 20 cubic centimetres at 0 deg. C. (32 degs. Fahr.) and 760 millimetres (29·92 inches), and was found to consist of the following:—

TABLE XVII.

	Volumes.
Carbon dioxide (CO_2)	0·0
Oxygen (O_2)	6·2
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0 0
Paraffins (C_nH_{2n+2})	21·4
Nitrogen (N_2)	72·4
	100·0

The hydrocarbons present gave results corresponding very nearly to propane (C_3H_8), the value for n being 3·08.

After this volume of gas had been removed, the coal was heated at 100 degs. C. (212 degs. Fahr.) for 12 hours, and 60 cubic centimetres more of gas was expelled. The total volume of gas obtained represents 44·44 cubic centimetres for every 100 grammes of coal. The last fraction of gas was found to have the following composition, the value found for n being in this case 2·15:—

TABLE XVIII.

	Volumes.
Carbon dioxide (CO_2)	2·8
Oxygen (O_2)	1·0
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	64·1
Nitrogen (N_2)	32·1
	100·0

Coal-dust from the same Colliery.

231 grammes of this coal-dust were heated at 100 degs. C. *in vacuo* for 12 hours; in this manner 50 cubic centimetres of gas were obtained. The gas proved to consist of—

TABLE XIX.

	Volumes.
Carbon dioxide (CO_2)	56·3
Oxygen (O_2)	7·1
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	8·5
Nitrogen (N_2)	28·1
	100·0

The combustible gas is similar in nature to that obtained from the Hebburn coal-dust, the value found for n in the formula, C_nH_{2n+2} being 3·22.

Backworth Colliery.

In connexion with the results of the examination of these several samples of coal, etc., taken from the hutton seam, county Durham, it may be well to state the outcome of an examination of a coal-dust representing the low main seam, Backworth colliery, Northumberland: 48 grammes of this coal-dust heated *in vacuo* at 100 degs. C. (212 degs. Fahr.) for 37·75 hours, gave off 18 cubic centimetres of gas and 5 cubic centimetres of water. The volume of gas represents 58·3 cubic centimetres for every 100 grammes of coal. The gas proved to have the following composition, and to contain hydrocarbons similar to those found in the Ryhope coal-dust, the value found for n being 2·1 :—

Table XX.

	Volumes.
Carbon dioxide (CO_2)	56·5
Oxygen (O_2)	14·5
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	18·5
Nitrogen (N_2)	10·5
	100·0

A consideration of the results obtained in the examination of these three different samples of coal, shows that they yield similar quantities of enclosed gases, and the combustible constituents form from 55 to 85 per cent. of these gases. Further, the combustible gases are very similar in composition, consisting of mixtures of the first members of the paraffin series of hydrocarbons, possibly with hydrogen in some cases. Again, these coals all yield dust, the enclosed gases of which will be seen to resemble one another in composition, and to contain combustible gases which are denser than those found in the coal.

These results have an interest beyond the demonstration of the fact that the denser hydrocarbons are more firmly held by the coal-substance than the lighter marsh gas, and that in the working of the coal the reduction to a fine powder, such as coal-dust, must by free exposure to the air allow ample opportunity for a ready diffusion of the marsh gas.

The existence of such enclosed gases in coal-dust show that, in considering the part played by coal-dust in forming explosive mixtures with air, we may have to deal with a variety of dust which might readily yield hydrocarbons similar to those found in the dust from these coals and that from the low main coal. These hydrocarbons differ from marsh gas, not only in the proportion required to form an explosive mixture

with air; thus whilst 1 volume of marsh gas would require 10 volumes of air for its complete combustion, 1 volume of the hydrocarbons found in the Ryhope coal-dust would need 23 volumes of air. In other words, whilst 9·3 per cent. of marsh gas is required to form the most explosive mixture with air, about 4 per cent. only of the combustible gases of the Ryhope dust would be required. The difference between marsh gas and the denser members of the series is not alone in the proportion of air that they require for complete combustion, but possibly also in the temperature of ignition of the explosive mixtures of air and these gases, for it has been recently demonstrated by Prof. Victor Meyer that explosive mixtures of oxygen and ethane ignite at lower temperatures than similar mixtures of marsh gas and oxygen.

These facts point to the ready inflammability and the sensitiveness of such dusts—a conclusion borne out by the ignition of the coal-dust on the screens at the Ryhope colliery—an occurrence which led originally to the investigation of this subject. Further, the Austrian Fire-damp Commission has, from the experiments with coal-dust, been led to the conclusion that the sensitiveness of dusts increases with the proportion of easily inflammable hydrocarbons, especially with the amount of ethane liberated at 100 degs. C. (212 degs. Fahr.), and with the dryness of the dust. There are doubtless other factors which contribute to the ready inflammability and sensitiveness of a coal-dust, and it can be readily understood how fineness of subdivision will contribute to make a dust easily inflammable. In addition dust which by long exposure to air must have absorbed a considerable volume of oxygen (this appears to be the case with the Brancepeth coal-dust) must necessarily be easily ignited, and may lead to violent and explosive combustion.

As bearing upon this property of the coal-substance to retain the denser hydrocarbons of the paraffin series, it may be interesting to cite here the results of the analysis of a sample of blower gas collected in the Hebburn colliery in October, 1891. The gas was found issuing from fissures in the rock and bubbling up through water 2 feet or so in depth. The gas was found to have the following composition, and the combustible gas to consist of marsh gas only:—

TABLE XXI.

	Volumes.
Carbon dioxide (CO_2)	0·70
Oxygen (O_2)	0·73
Carbon monoxide (CO)	0·00
Olefines (C_nH_{2n})	0·00
Marsh gas (CH_4)	83·67
Nitrogen (N_2)	14 90
	100·00

On the occasion on which this sample was collected, another was taken of the gas, which is piped to bank at this colliery. The composition of this gas was found to be as follows :—

TABLE XXII.

	Volumes.
Carbon dioxide (CO_2)	0·9
Oxygen (O_2)	1·7
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Marsh gas (CH_4)	78·8
Nitrogen (N_2)	18·6
	100·0

The absence from the fire-damp of the higher members of the paraffin series, or their presence in such small amounts as not to show in the analysis, supports the views expressed above as to the firm retention by the coal of the denser hydrocarbons.

BACKWORTH COLLIERY.

Two other varieties of coal-dust have been examined—viz., dust from the coal of the bensham seam, at the Backworth colliery, Northumberland, and the dust collected on the Hartley screens of the Seaton Delaval colliery, Northumberland, the latter representing the yard seam coal. The first of these coal-dusts gave a volume of gas, equivalent to 45·4 cubic centimetres per 100 grammes of coal-dust. The gas was found to have the composition given below, and to contain, apparently, some ethane in addition to marsh gas. It will be seen to resemble the gas obtained from the low main coal.

TABLE XXIII.

	Volumes.
Carbon dioxide (CO_2)	35·1
Oxygen (O_2)	4·4
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	10·1
Nitrogen (N_2)	50·4
	100·0

SEATON DELAVAL COLLIERY.

The coal-dust from the yard seam at Seaton Delaval has been examined with the results stated below, which show that this coal-dust contains no combustible gases, and further that the dust resembles very nearly in the character and amount of its enclosed gases the dust collected in the hopper at Brancepeth :—

TABLE XXIV.

Samples.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Water in Cubic Centimetres.	Volume of Gas reduced to Volume at 0° C. and 760 mm. Mercury Pressure.
				Cubic Centimetres.
A	4	40°–60° C. (104°–140° Fahr.)	—	324
B	17	50°–70° C. (122°–158° Fahr.)	—	1,045
C	2	60° C. (140° Fahr.)	—	316
D	—	—	—	—
E	45	100° C. (212° Fahr.)	25	305
F	17	100° C. (212° Fahr.)	10	10
Total ...	85	—	35	2,000

And consequently 100 grammes of coal-dust yielded 42·21 cubic centimetres of gas.

The several samples on analysis gave the following results :—

TABLE XXV.

Samples.	A.	B.	C.	E.	F.
Carbon dioxide (CO_2) ...	6·4	4·0	8·4	32·3	74·1
Oxygen (O_2)	10·7	10·0	6·3	0·5	0·0
Carbon monoxide (CO) ...	0·0	0·0	0·0	0·0	0·0
Olefines (C_nH_{2n})	0·0	0·0	0·0	0·0	0·0
Paraffins (C_nH_{2n+2})	0·0	0·0	0·0	0·0	0·0
Nitrogen (N_2)	82·9	86·0	85·3	67·2	25·9
	100·0	100·0	100·0	100·0	100·0

The above results may be expressed as follows :—

TABLE XXVI.

Samples.	Volume in Cubic Centimetres.	Carbon Dioxide. CO_2.	Oxygen. O_2.	Nitrogen. N_2.
A	324	20·7	34·7	268·6
B	1,045	41·8	104·5	898·7
C	316	26·6	19·9	269·5
D	—	10·0	—	—
E	305	98·7	1·5	204·8
F	10	7·5	0·0	2·5
	2,000	205·3	160·6	1,644·1

The average composition of the gas would therefore be the following :—

TABLE XXVII.

	Volume per Cent.
Carbon dioxide	10·26
Oxygen	8·03
Nitrogen	82·20
	100·49

The coal-dust during the extraction of the gases lost in weight 58·5 grammes.

Further Experiments with Seaton Delaval Coal-dust.

The weight of coal-dust taken was 739 grammes. The coal-dust was heated *in vacuo* at the following temperatures and gave :—

TABLE XXVIII.

Samples.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Water.	Volume of Gas reduced to Volume at 0° C. and 760 mm. Mercury Pressure.
			Cubic Centimetres.	Cubic Centimetres.
A	11	80° C. (176° Fahr.)	—	509
B	5	85° C. (185° Fahr.)	5	484
C	13	85° C. (185° Fahr.)	—	485
D	20	100° C. (212° Fahr.)	55	40
E	35	100° C. (212° Fahr.)	—	10
	84	—	60	1,528

Consequently 100 grammes of coal-dust yielded 20 cubic centimetres of gas.

The analysis of the several samples yielded the following results :—

TABLE XXIX.

Samples.	A.	B.	C.	D.	E.
Carbon dioxide (CO_2) ...	19·0	11·0	19·9	85·0	9·0
Oxygen (O_2) ...	3·5	16·6	0·0	0·0	0·0
Carbon monoxide (CO) ...	0·0	0·0	0·0	0·0	0·0
Olefines (C_nH_{2n})	0·0	0·0	0·0	0·0	0·0
Paraffins (C_nH_{2n+2}) ...	0·0	0·0	0·0	0·0	0·0
Nitrogen	77·5	72·4	80·1	15·0	10·0
	100·0	100·0	100·0	100·0	100·0

The above results may be expressed as follows :—

TABLE XXX.

Samples.	Volume in Cubic Centimetres.	Carbon Dioxide. CO_2.	Oxygen. O_2.	Nitrogen. N_2.
A	509	102·8	17·3	389·4
B	484	53·2	80·6	350·4
C	485	97·6	0·0	388·0
D	40	34·0	0·0	5·4
E	10	9·0	0·0	1·0
	1,528	296·6	97·9	1,134·2

The average composition of the gas obtained in this series of experiments is therefore :—

TABLE XXXI.

	Volumes per Cent.
Carbon dioxide	19·4
Oxygen	6·4
Nitrogen	74·2
	100·0

The coal-dust lost 75 grammes in weight during the extraction of the gases.

Experiments with Seaton Delaval Coal and Crushed Coal.

To institute a comparison between the coal and its dust a similar series of experiments was made with this coal. The weight of coal taken was 637 grammes, and this was heated *in vacuo* at the following temperatures, yielding these results :—

TABLE XXXII.

Samples.	Number of Hours Coal was Heated.	Temperature of Bath.	Volume of Water.	Volume of Gas reduced to Volume at 0° C. and 760 mm. Mercury Pressure.
			Cubic Centimetres.	Cubic Centimetres.
A	5	50°–80° C. (122°–176° Fahr.)	10	818 = sample A.
A′	9	100° C. (212° Fahr.)		
B	4	50°–100° C. (122°–212° Fahr.)	50	1,052 = ,, B.
B′	14	100° C. (212° Fahr.)		
	32		60	1,870

Consequently 100 grammes of coal yielded 29 cubic centimetres of gas. These samples on analysis gave the following results :—

TABLE XXXIII.

	Sample A.	Sample B.
Carbon dioxide (CO_2) ...	13·2	19·0
Oxygen (O_2)	5·9	1·0
Carbon monoxide (CO) ...	0·0	0·0
Olefines (C_nH_{2n})	0·0	0·0
Paraffins (C_nH_{2n+2}) ...	0·0	0·0
Nitrogen (N_2)	80·9	80·0
	100·0	100·0

The above results may be stated thus:—

TABLE XXXIV.

Samples.	Volume of Gas.	Carbon Dioxide (CO_2).	Oxygen (O_2).	Nitrogen (N_2).
A	818	106·9	48·3	661·8
B	1,052	199·9	10·5	841·6
	1,870	306·8	58·8	1,503·4

The average composition of the gas yielded by this coal would therefore be:—

TABLE XXXV.

	Volumes.
Carbon dioxide	16·4
Oxygen	3·2
Nitrogen	80·4
	100·0

Crushed Coal.

The coal, after it had ceased to give off gas, under the conditions already described, was crushed to a fine powder, and again submitted to examination. The ground coal, the weight taken being 532 grammes, was heated *in vacuo* at 100 degs. C. (212 degs. Fahr.) for 10 hours, and gave 494 cubic centimetres of gas and 20 cubic centimetres of water. 100 grammes of ground coal yielded 9·2 cubic centimetres of gas.

The analysis of this sample gave the following results:—

TABLE XXXVI.

	Volumes.
Carbon dioxide (CO_2)	16·5
Oxygen (O_2)	7·0
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Paraffins (C_nH_{2n+2})	0·0
Nitrogen (N_2)	76·5
	100·0

With the object of obtaining further information on the manner in which some of these gases are retained by the coal-substance a series of experiments have been made with ethane, charcoal, and coal, and a second series in which the behaviour of coal in an atmosphere of hydrogen has been studied.

The results are detailed below.

Retention of Ethane by Charcoal and Coal.

It appeared desirable to determine whether either charcoal or coal, from which the enclosed gases had been previously removed by heating *in vacuo* at 100 degs. C. (212 degs. Fahr.) possesses any power of absorbing and retaining the hydrocarbon ethane. To this end a given weight of boxwood charcoal was introduced into a glass tube (sealed at one end) 10 centimetres wide, the other end of which was drawn out and fused on to the air-pump. The air from the tube, and that retained by the charcoal, were carefully pumped out, and then the pump was filled with ethane, prepared by the action of zinc ethide upon water, the gas produced being washed by passing through (*a*) an alcoholic solution of soda, (*b*) bromine dissolved in a solution of potassium bromide, and (*c*) caustic potash. The charcoal, thus surrounded by an atmosphere of ethane, was left for several days under a pressure of some 1,460 millimetres of mercury. At the end of this time the gas remaining unabsorbed was removed, and the pump exhausted completely.

The tube containing the charcoal was then heated by passing steam through a jacket surrounding it, and the gas thus liberated passed over into the pump, and was drawn off, its volume measured, and its composition determined. From the composition of the gas expelled the volume of ethane absorbed by the charcoal was deduced.

The ethane used in these experiments, and prepared in the manner described above, was analysed and found to have the following composition :—

Table XXXVII.

	Volumes.
Carbon dioxide (CO_2)	0·0
Oxygen (O_2)	2·8
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	1·2
Nitrogen (N_2)	6·8
Ethane (C_2H_6)	89·2
	100·0

The gas expelled from the charcoal by heating at 100 degs. C. (212 degs. Fahr.) *in vacuo*, measured 70 cubic centimetres, and was proved by analysis to consist of :—

Table XXXVIII.

	Volumes.
Carbon dioxide (CO_2)	0·0
Oxygen (O_2)	1·6
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n})	0·0
Nitrogen (N_2)	69·3
Ethane (C_2H_6)	29·1
	100·0

From this it is seen that the charcoal taken, viz., 18·5 grammes, retained, at the ordinary temperature and under reduced pressure, 20·23 cubic centimetres of ethane, which were only expelled when the charcoal was heated to 100 degs. C. In other words, 100 grammes of boxwood charcoal, when saturated with ethane, and then placed *in vacuo* at ordinary temperature, is still capable of yielding 109·3 cubic centimetres of ethane, measured at 0 degs. C., and 760 millimetres, when heated at 100 degs. C.

A similar series of experiments were made with Hebburn coal. For this purpose 114 grammes of this coal were introduced into a tube sealed at one end and drawn out at the other end, by which it was sealed on to the air-pump. After the pump had been exhausted, the coal was heated by passing steam through the jacket surrounding the tube, and the heating continued for 68 hours. Gas having ceased to come off from the coal, the gas was completely removed, and then the pump filled with ethane. The pressure on the gas in the pump was increased to 1,500 millimetres of mercury, and under these conditions the coal was left surrounded by an atmosphere of ethane for 77 days. At the end of this time the pump was exhausted, and then the tube containing the coal was heated at 100 degs. C. for 29 hours. In this manner 126 cubic centimetres of gas, measured at 0 degs. C. and 760 millimetres was obtained, the analysis of which showed it to consist of the following:—

TABLE XXXIX.

	Volumes.
Carbon dioxide (CO_2)	0·0
Oxygen (O_2)	0·0
Carbon monoxide (CO)	0·0
Olefines (C_nH_{2n}	0·0
Ethane (C_2H_6)	96·0
Nitrogen (N_2)	4·0
	100·0

From this it is seen that coal absorbs ethane, and that 100 grammes of coal will retain 96·48 cubic centimetres of ethane. There is, therefore, but little difference between charcoal and coal in their behaviour towards ethane.

EXPULSION OF GASES FROM COAL BY DIFFUSION.

To ascertain whether it would be possible to effect a liberation of the enclosed gases from coal, by means of diffusion into an atmosphere of a lighter gas, the following experiments were made:—

A given weight of Ryhope coal was introduced into a round-bottomed flask, the neck of which was closed by a two-holed indiarubber stopper. In one of these holes a long glass delivery-tube was inserted, the open end of which was fixed under mercury in a pneumatic trough placed some 30 inches below the flask. Into the second hole in the stopper was inserted a glass tube which passed through the coal to the bottom of the flask. Hydrogen was led through this second tube to displace the air and fill the flask entirely with this gas. The gas issuing from the delivery-tube was collected from time to time, and when analysis proved it to be pure hydrogen, the tube leading the gas into the flask was closed. The coal was then left for some hours surrounded by hydrogen, and then a sample of gas was drawn off by attaching a mercury pump to the delivery-tube. The flask was then heated in a water-bath, and a further quantity of gas was obtained. The analyses of these first samples showed the gas to be free from hydrogen, and to contain oxygen and nitrogen. Imagining that this might have been due to leakage, a second experiment was made, but with apparatus constructed entirely of glass. For this purpose a distilling flask, provided with a lateral tube, was taken and filled with the coal; into the mass of coal, and passing through to the bottom of the flask, a glass tube was inserted, and the neck of the flask was drawn out and sealed round this tube. A piece of glass-tubing was sealed on to the lateral tube of the flask to form a long delivery-tube, the open end of which dipped under mercury in a pneumatic trough.

The flask was next filled with hydrogen by passing a current of this gas through the tube in the neck of the flask and outwards through the opening of the long delivery-tube. When the gas issuing from this tube was found to consist of hydrogen alone, and all the air had been expelled, the portion of the tube projecting beyond the neck of the flask was sealed off at the blowpipe. In this manner the coal was surrounded by an atmosphere of hydrogen, and maintained under these conditions for some hours. The flask was then heated by a water-bath, and the gas liberated collected by displacement of mercury from tubes placed over the delivery-tube in the pneumatic trough.

The following table contains a summary of the analyses of a series of samples collected in this manner :—

TABLE XL.

Sample.. .. .. Temperature of extraction ..	A. 100° C.	B. 100° C.	C. 100° C.	D. 100° C.	E. 100° C.
Carbon dioxide (CO_2) ...	0·0	2·00	2·30	3·30	4·00
Oxygen (O_2)..	1·2	1·90	0·00	0·00	0·00
Carbon monoxide (CO) ...	0·0	0·00	0·00	0·00	0·00
Olefines (C_nH_{2n})	0·0	0·00	0·00	0·00	0·00
Hydrogen (H_2)	0·0	88·83	67·91	73·40	60·77
Marsh gas (CH_4)	0·0	6·05	13·19	16·44	20·16
Nitrogen (N_2)	98·8	1·22	16·00	6·86	15·07
	100·0	100·00	100·00	100·00	100·00

The explosion with oxygen of the gases left after the removal of oxygen and carbon dioxide gave the results contained in the following table, these numbers representing volumes expressed in terms of ten volumes of the residual gas :—

TABLE XLI.

Sample.	A.	B.	C.	D.	E.
Contraction	—	13·70	14·40	14·80	19·00
Carbon dioxide ...	—	0·63	1·35	1·70	2·10
Oxygen used	—	5·56	7·48	6·29	12·60
Nitrogen	—	0·13	1·70	0·71	1·57
Hydrogen	—	9·24	6·95	7·59	6·33
Marsh gas	—	0·63	1·35	1·70	2·10
Total contraction ...	—	14·33	15·75	16·50	21·10

It is evident, therefore, that an exchange of gases takes place between the coal and the hydrogen atmosphere, the coal at first taking up the hydrogen, which in turn expels nitrogen and a little oxygen from the coal. Then as the heating is continued the hydrogen is expelled, and along with it some marsh gas, the proportion of the latter and of the carbon dioxide increasing as the heating is continued and the pressure reduced consequent on the expulsion of gas from the apparatus. These results demonstrate not only the possibility of expelling the gases from coal by diffusion, but also the fact that the coal contains nitrogen enclosed in it, not simply as air, but apart from oxygen.

From these investigations it is evident that the coals of the Durham and Northumberland coal-fields exhibit wide differences in the nature of their enclosed gases; whether these differences are in any way related to their geological position it would be rash to conclude from the data provided, the question requiring more extended investigation.

The material before us would at any rate show that the coal of certain seams produces in working, dusts which might readily lend themselves to inflammation from the nature of their enclosed gases, whilst others may prove easily inflammable by reason of the absorbed air which they contain.

In conclusion, the authors wish to express their thanks to the several proprietors and managers of the collieries from which the coals, etc., have been obtained, for their kindness in rendering assistance by the supply of material. It has thus been possible to conduct investigations which the authors hope may be considered as forming a practical contribution to our knowledge of coal-dust.

Mr. W. C. Blackett read the following paper on "The Combustion of Oxygen and Coal-dust in Mines":—

THE COMBUSTION OF OXYGEN AND COAL-DUST IN MINES.

By W. C. BLACKETT.

The subject of coal-dust has of late years occupied the minds of so many people, and proved to be of such paramount importance to the safe-conduct of coal-mines that too much cannot be said about it, even if the saying necessarily entails what to many will be wearisome repetition. This is the writer's excuse for bringing forward some notes on the subject, and he trusts that those to whom it may be somewhat tedious will give him as much consideration as possible in an effort which may possibly bring the matter more clearly before the minds of others who either have not sufficiently studied it, or in doing so have been confused by the various lights in which the many writers have seen those phenomena which they have sought to describe.

To those who wish to pursue the study more closely the writer cannot do better than recommend a careful perusal of the book of the Messrs. Atkinson.* Nothing better has been written, and he feels bound to add his testimony to the clear and comprehensive grasp which they have of the subject.

The Final Report of the Commission appointed to enquire into Accidents in Mines gives an historical account of nearly all that was known upon the matter up to 1886, and of the various experiments that had been carried out, while the First Report of the Coal-dust Commission of 1891 will bring the subject fairly up to date.

The writer specially speaks of combustion rather than explosion, as he finds that the use of this latter term by a great number of people produces quite a different train of thought to that of the idea of combustion. He attaches the greatest importance to this primary definition. To his mind no difference, except in degrees of force and speed, exists in the phenomena variously described as combustion, explosion, ignition, inflammation, detonation, and the like. He is aware that this latter term is somewhat hesitatingly used by many people, possessing as it does

* *Explosions in Coal-mines.*

characteristics peculiarly its own, but differing mysteriously from those of explosion. In fact, however, they should all come under one heading, and are all the result of chemical re-combination, differing, however, in their physical causes and effects.

In certain instances it seems as though sympathy of vibration were required before a substance could effect its chemical re-combination, and as though the degree of effect were proportional to that of the cause. The sounding by one body of a musical note sets up in another body in sympathy with it a similar number of vibrations, and, consequently, a similar note, and it may be that to some such property certain high explosives owe their safety. The vibrations of an ordinary flame are not sufficient to explode, say, ammonite or roburite, which require more harmonious treatment.

Not until there is a perfect understanding of the relations to each other of these various matters—detonators, explosives, gases, dusts, and what not—their chemical and physical properties, can we expect to understand, as we ought to do, explosions in mines. We look forward eagerly to the results of all experiments, and at the present time more especially to those initated by the self-sacrificing labour of the very able committee of this Institute.

Introduce hydrogen gas to oxygen in the right proportions; allow it time to intermix by its physical property of diffusion, and on applying a light you will get a violent ignition. Introduce, and at once fire, the gas, however, as it were in bulk, and the ignition would not be so violent. Similarly, introduce coal fine enough—diffused enough, the result would be combustion more or less violent. In the lump, however, it would but burn as in our fire-places.

No one will deny that dust exists in collieries in form and in place peculiarly favourable to being disseminated more or less easily throughout the passing air-current. It has the property of being able to deposit itself on almost any surface in festoon-like masses, quite regardless of the speed of the air-current.

If all could agree upon one simple way of looking at the matter the difficulties would soon be overcome. It is extraordinary to find what a great number of points of view there are shown to be in the evidence taken before the recent Royal Commission. One eminent authority is teaching that dust must be mixed with air so thickly that one cannot see through it before the mixture becomes explosive. Another asserts that a quantity of dust barely sufficient to soil a white handker-

chief is the more dangerous; another that there must be, in addition to the dust and air, a small percentage of fire-damp; while yet another declares that dust and air cannot be exploded, but only ignited.

The writer had opportunities of partially examining the state of the workings at Seaham, Tudhoe, Trimdon Grange, and Usworth collieries, after their disastrous explosions, and on the facts there observed has formed a theory of the action of dust and air-ignition, which may be briefly summarized as follows:—

Firstly, there is an initiatory force, a gunpowder shot, a local fire-damp explosion, or perhaps a heavy fall of roof. This stirs up a thick cloud of dust, which can be immediately ignited by any accompanying flame. At this stage no great violence is developed—merely an expanding inflammation, producing, however, an onrushing cloud of heat-dilated air and dust, and for 70 or 80 yards no force is produced sufficient to damage or move light articles; the men there at work are only burnt; their drilling gear, etc., is found upstanding. But the cloud is now advancing against greater difficulties. In one direction it had to overcome the momentum of a swift air-current and the inertia of a mighty mass of air beyond; in the other direction it had this swift current first to overtake, and then with its terrible speed to overcome an inertia, although not so great, yet for its purpose almost equally so. The former is stayed and reversed, and both are whirled away at an ever-increasing pressure, varying, as we have all learnt, with the square of the velocity. If the shaft in one direction, and the expanding face of workings in the other are a long way off, so much the worse will be the effects.

The increasing commotion is stirring up greater clouds, and the increasing pressure is serving to intimately mix therewith more abundant oxygen, and pursuing this increasingly concentrated explosive mixture is the flame. Air is, however, so elastic that this accelerated movement will, in the first instance, have been quite gentle, and in its flow it would naturally follow those paths most easy for it—those, indeed, that it would have followed had it been guided by the ordinary laws of ventilation. Out-bye it takes the nearest open way to the surface, and in-bye to the expanding area of workings. This gentle flow prepares the way for the rush; the rush stirs up the dust and pioneers the path for the oncoming explosion. There is no turning to the right or left, and no going through doors or stoppings into the returns; but as the explosion passes down go these obstacles, beyond which, however, the road is not prepared

and therefore the explosion does not extend therein. The shaft is soon reached, the pressure is relieved, and the explosion ceases. Or perhaps up the shaft the velocity and counterweight are so great that the pressure can be actually higher than the resistance of a road beyond. In this case the pioneering phenomena are continued, and the explosion progresses.

In the direction of the face the explosion has proceeded until the greater number of open galleries has reduced the speed of the rushing air, so that it will no longer produce a thick enough cloud of dust, and it dies away. It has perhaps, indeed, partly provided its own safety-valve, for the stoppings and doors into the returns have been blown down by the passing explosion.

All the available oxygen has been consumed: indeed there has not always been sufficient for complete combustion, and instead of carbon dioxide alone, the deadly carbon monoxide is also found.

If a damp part of the road sufficiently long, or one free from dust is encountered, of course the pioneering work is incomplete and the explosion ceases. Similarly, if the area of the road were sufficiently increased, it would probably also cease.

The writer has not yet noted at any colliery explosion one single fact that is not completely in accord with this conception, which gives very good reasons why an explosion does not go into the returns. Of course he fully recognizes that there is more dust in the intake, but still he believes that if an explosion got fairly started in some returns the result would be just as disastrous.

It should not be forgotten that this dust, which is sometimes so thick in the intakes, may in its deposit build up a structure favourable to absorption of some inflammable gas, while Dr. Bedson has conclusively proved that the particles themselves have this power.

As to the quantity of dust. It may be quite true that theoretically the volume of dust proportionate to that of air necessary to produce correct chemical results is very small, but practically a great deal more dust is necessary to produce a colliery explosion. No one, the writer supposes, would contend that a large block of coal in air would be even easy to burn. If, however, it is divided into smaller parts it will become correspondingly easier, but never so easy as it would be when finally reduced to the molecules of gas. But it cannot be reduced to such molecules and retain its form of dust, and the finest dust produced cannot in this respect be compared with molecules.

With gas, however, it is different. Molecules of fire-damp can be in contact with those of oxygen ready for the change to be produced by flame, and the theoretical conditions are fulfilled.

If the dust were distributed in theoretical quantities evenly throughout the air, one little grain would be too far off another to transmit heat to carry on the combustion. But if a greater number of grains are put nearer each other, the heat will be easily transmitted, and although the grains will not be fully consumed, yet the oxygen would take all it wanted or could do with and leave the rest. This forms a sufficient answer to the suggestion that dust in air so invisible as not to soil a handkerchief could explode. It is undoubtedly a dangerous form of dust, and would probably do for its size the most of the work in an explosion, but it would have to be largely helped.

It is perhaps fortunate that dust cannot be ignited in its theoretical proportions with air, for then the violence would be greater. Surely the rendering incandescent of a large amount of unnecessary dust must absorb an amount of energy that might otherwise be exerted to swell the explosive force.

Perhaps no grain of dust is ever completely consumed. Probably the oxygen is all used up by the more volatile hydrocarbons and none is left for the slower burning carbon.

Instances like the ignition in the Brancepeth hopper show that a mere mixture of dust and air will ignite and burn fiercely. Mr. R. L. Weeks gave evidence upon this accident before the Royal Commission and drew an undefined distinction between inflammation and explosion, admitting the former but denying the latter. His evidence established three things:— Firstly, that the dust in the hopper was in a cloud so thick that it could not be seen through or breathed; secondly, that with an open light alone there was ignition and inflammation; and thirdly, that there was not much violence. Here we have an exact parallel to what takes place in the mine. The dust thickly stirred up by a heavy shot, the ignition by the flame thereof, and the absence of any great violence. Then follow the subsequent phenomena of air-compression and violence. If instead of one atmospheric pressure in the hopper there had been two, will it be admitted that then there would have been sufficient violence as to have received the name of explosion?

When does combustion change to explosion? Is there a boundary-line between the two? In the ignition at Brancepeth one nailed sheet was, to quote the evidence, "burst off the top." Would two sheets or three so damaged make it worthy of being called an explosion?

The thickness of the cloud was important. Of course the oxygen in the air only really united with its theoretical amount of hydrocarbon and carbon, but it took a great deal more than that amount to make it do so. In connexion with this the author draws attention to Mr. Dixon's paper on "The Rate of Explosions in Gases,"* where he points out that the results of his experiments are in favour of the view that in the explosion of marsh gas, etc., the carbon is first burnt to carbonic oxide gas. His paper is further interesting in that it seeks to establish that between certain gases there are specific constants in the rates of velocity of explosion, which seems to have some connexion with the rate of the molecules of the exploding gases.

How far this law would govern explosions of oxygen and coal-dust it is difficult to say, but if ever the rate of oxygen and fire-damp could be reached, it is said that we should have a velocity of 458,640 feet per minute! It does not seem probable that this law affects the ignition of coal-dust to any great extent, and the writer cannot agree with Mr. Dixon when he says that similar phenomena as studied by Messrs. Mallard and Le Chatelier are at all parallel with those "observed on a large scale in explosions in coal-mines."

As to the amount of compression in the air preceding the ignition, surely anyone can call up a mental picture of what this may be. If there be any difficulty, then let a simple experiment be tried. Fix a water-gauge to a separation-door, and having opened the latter for a sufficient time to allow of the establishment of the air-current, shut it suddenly and observe the behaviour of the gauge. Then imagine, instead of the mere stoppage of such a current, its sudden reversal at a high velocity, and it may be that some idea of the possibility of the production of high pressure may be obtained.

It has been shown by Sir Frederick Abel and others that the physical properties of dust may be as important as the chemical, and it would appear as if certain dusts acted as conductors either of heat or of some other form of energy. Thus it was found that magnesia in a state of fine suspension helped to explode an otherwise non-explosive mixture of fire-damp and air, and, in connexion with this, Sir Frederick Abel said that such physically peculiar dust exerted a distinctly catalytic action upon gas mixtures. It seems therefore quite possible that a similar action may be exerted in dusts during their own combustion with air.

* *Trans. Fed. Inst.*, vol. iii., page 312.

Dr. Bedson, in his papers to this Institute, has shown that coal-dust encloses many times its own volume of very combustible gases more or less easily liberated, and the importance of this discovery cannot be exaggerated. Mr. W. E. Garforth, in his evidence before the Royal Coal-dust Commission, may therefore be right in surmising that the spore-envelopes which he saw in exploded coal-dust under the microscope might contain gas at considerable pressure. The spores, too, may themselves be more highly inflammable.

In emphasizing the claim that coal-dust alone and air can carry on a violent ignition amounting in force exerted to what is generally called explosion, the Royal Commission on Accidents in Mines ask how it is that every blown-out shot occurring in dusty mines is not attended by more or less disastrous explosion or conflagration? The answer is, that there are so many conditions to be exactly fulfilled, that although each one is common, yet the whole combined are fortunately rare and difficult to attain. There must be enough spare power in the shot to stir up the dust and produce enough compression of air. The dust must be very fine and dry; the flame from the explosive must be of very high temperature and considerable volume. Possibly the shape of the gallery, the relative temperature of the air, and the height of the barometer may also be important. Taken altogether, the chances of the complete coincidence are so small that we need not wonder at the rarity of the accident.

If it is difficult to produce these conditions underground, how much more must it be so in experiments on the surface. The first experimentalists, such as Messrs. Abel, Mallard, and Le Chatelier, and many others found it difficult to get violent combustion without adding that always very necessary and never neglected small percentage of fire-damp. These defective experiments have delayed the proper appreciation of coal-dust more than anything else. When explosions occurred the question always arose how was this small quantity of gas, which experimentalists had declared was necessary, to be accounted for? They said it must have been there, and there it had to be. Perhaps one of the most interesting suggestions was that known as the suction theory. To illustrate it the action of a cannon was instanced, placed at the closed end of the boiler and fired in the direction of its mouth. A tube was inserted just above the cannon, and the collapse of this tube upon the firing of the cannon was very properly held to show that a partial vacuum had been produced. But where is the parallel in this to an explosion in a coal-mine? Any parallel there might be is with the cannon itself and what takes place therein, which would hardly be held to constitute a vacuum.

It is true that after the blast has passed by, and the temperature of the heated gases begins to decrease there will be returning currents, often amounting in speed perhaps to that necessary for those phenomena observed and accounted for by the Messrs. Atkinson and others and named back-lash. It does not follow, however, that this back-lash should always return along the path of the explosion. Thus, as the explosion passes and knocks down a stopping, it opens for itself a communication whereby not only the condensation of the heated contents could be satisfied, but also whereby the rush of air *via* the working-face and return in front of the explosion could find vent, and hence in some cases there might actually be a circulating path of phenomena. Or again, the pressure arising from the blast might penetrate through the stoppings and prevent the continuance of the explosive phenomena. In that case there might be air compressed, say at the face, but in a comparative state of rest. Then the explosion, for want of pioneering phenomena, would die away.

The volume of air driven out of both upcast and downcast shafts by the explosion, should be equal to the total volume of all the back-lash, and the writer expects to find it re-entering by both shafts. Has anyone noticed whether this is so ?

The shape of the mine and the variety of forces set in motion make it very important that the explorer should not readily jump to conclusions in seeking for indications of the direction of the blast. The writer remembers at Seaham colliery how eagerly everyone examined the position of the coal-dust on props and other objects. Now, nothing could well be more deceptive. Even such careful observers as the Messrs. Atkinson find it difficult to account for all the vagaries of dust. As an instance of this in their examination of Usworth colliery they found that " the stoppings on the left side going in-bye were set close to the intake airway ; on the right side they were set back some yards and the space between them and the intake was used as a deposit-hole for dust and rubbish. This dust and rubbish was in some cases strewn in-bye. There could be no doubt that the force of the explosion travelled out-bye past these points. Air would be forced into these places against the stoppings, and the recoil would appear to have brought out some dust and rubbish, and from the direction in which it was strewn the air would appear to have reversed." *

Now, this is exactly what might have been expected. The in-bye portion of the refuse hole would be filled with compressed gases, and, after

* *Explosions in Coal-mines*, page 76.

the passage of the blast, these would expand and flow out in the direction of least resistance, which would first be away from the blast, travelling to the shaft towards the cooling and condensing gases left behind, and the dust would, of course, be carried therewith and deposited as shown.

To be able to say exactly how the dust has come into a certain position after a blast, one must know its position beforehand. It is deposited on two sides of timber in mines, that against the air-current, and upon the sheltered side to which it would appear to have sunk in an eddy. Snow can be seen to do this when driven by the wind past a gate or the like. We have, therefore, before an explosion, dust on both sides of a prop. Now, two things can happen:—Either the blast will remove the dust from the exposed side, and the flame will lick round, and coke that on the sheltered side, or it can be removed by the blast from both sides, and redeposited in a semi-coked condition by the gentler and moister returning current. Either of these occurrences would account for many of the facts observed after an explosion.

Accepting the aforementioned views, there is also no longer any difficulty in accounting for the apparent vagary of the blast where it followed, as an instance, the haulage-road in the No. 1 pit at Seaham colliery, and not the adjoining travelling-way. It is no doubt a very powerful reason to give, to say that it was because the nature of the dust was more favourable in the former than the latter, but it is far more important that the former was the more direct road, and was better prepared than the latter. Then, again, even if the pioneering phenomena were set up at the entrance to the travelling-way at the outer end, they would be met and stopped to a large extent by an opposing entry at the other end. One would then expect to see the blast partially enter the travelling-way and proceed for a greater distance from the out-bye than from the in-bye end, owing to the interval of time consumed by the blast travelling from one end to the other on the haulage-road. In other words, had the travelling-way communicated with another district instead of returning again to the haulage-road, there would have been as great an explosion in the former as in the latter.

This same reason of balanced pressure may be found also to account for the non-entry of the blast into many other parts of the mine apparently open, because it prevents the occurrence of the necessary pioneering phenomena. A door or a stopping may have the same effect, as shown in the instance of the Lawson way at Usworth colliery.

It may be found impossible to make experiments producing results similar to those which obtain in the mine itself, but if it is to be done the

conditions of the mine must be followed as closely as possible. The experimental gallery should be made very long and the equivalent resistance to the friction of the mine produced on the air-current by the artifice of restricted orifices. Ventilation should be effected by a fan, and the air-current should be allowed to deposit the experimental dust. Blown-out shots should not necessarily be the only shots fired, but others should be fired which by an arrangement of heavy weights might be allowed to do partial work, and thus perhaps produce a more reverberating effect. An intake and return airway should be provided with stoppings and the like. A *cul-de-sac* also in which the comparative pressures might be easily gauged and the effect of varying the resistance or the sizes of the equivalent orifices observed. The air should be passed through some drying agent to produce the necessary hygrometrical condition, and the interior should be heated to the temperature of the mine by steam-pipes.

The writer has every hope that when this is done it will be found that none of the high explosives, such as roburite and ammonite, will fire coal-dust, but that only such slow explosives as gunpowder, which are in sympathy with dust-ignition will do so.

Mr. J. C. B. Hendy asked Messrs. Bedson and McConnell whether all their experiments were made on coal in a vacuum, or whether they had made any experiments which would be likely to show what gases would be evolved from coal and coal-dust under ordinary conditions; the conditions, for instance, of the coal at the face and of the coal-dust on the haulage-roads of a mine? They knew, that soft and porous coals evolved their gases more quickly than hard and close-grained coals. He believed also that different coals, although they might be of the same chemical composition, might differ entirely in their combustibility, and therefore, although high percentages of inflammable gases were found when heating certain coals in a vacuum, it might not follow in every instance that the dust from the same coal would be highly inflammable or explosive.

Mr. Thomas Bell (H.M. Inspector of Mines) asked Dr. Bedson what time elapsed, after the explosion in the hopper at Brancepeth colliery, before the sample of dust was taken?

Dr. Bedson said he thought the sample was taken about two days after the fire occurred.

Mr. R. L. WEEKS said the dust in the hopper accumulated very rapidly on the flat planks.

Mr. THOMAS BELL said the object of his question was to ascertain whether the samples of dust had been affected by the heat of the explosion. Was it dust that had been already burned?

Dr. BEDSON said the dust had not been burnt.

Mr. R. L. WEEKS said the hopper was washed out after the explosion.

Mr. W. C. BLACKETT said he had been much struck by one of Dr. Bedson's remarks, in which he explained the presence of carbon dioxide as being probably due to oxidation of some of the dust. Was it likely that, in the first place, the dust deposited in coal-mines contained the bulk of carbon dioxide which was afterwards expelled by other gases? Dr. Bedson referred to the absorption of carburetted hydrogen, would this carburetted hydrogen expel any of the carbon dioxide previously contained in the dust?

The PRESIDENT said that Dr. Bedson's paper contained a very valuable consideration of the coal-dust question and there was no doubt that before they could understand the subject, such questions as the constitution of the dust ought to be thoroughly examined. In Dr. Bedson's evidence before the Royal Commission on Explosions from Coal-dust in Mines* he explained more clearly what the value of his work was than perhaps he (the President) could do. Dr. Bedson then stated that the Ryhope dust stood out prominently from others as containing very considerable proportions of combustible gases, whilst the Brancepeth dust contained very little. He (the President) would suggest that though the Ryhope dust did contain inflammable constituents of the paraffin and olefine series, still they had the fact that the Brancepeth dust did explode from contact with a naked light although these highly explosive gases were not found in it. He would like to say a word to Mr. Weeks, who in his evidence before the Royal Commission went so far as to say "he did not think that an inflammation of coal-dust could ever be the exciting cause of an explosion,"† although he had an explosion from coal-dust and coal-dust alone in his hopper. The coal-dust being unconfined, was a sufficient reason for not having what he called an explosion. If they were to strew gunpowder on the floor it would not explode, but would burn away; but let them put it under pressure, as coal-dust was in a long length of airways, and they would get the necessary conditions for an explosion. Explosion was to a certain extent the result of being confined, and when the coal-dust took fire in the hopper at Brancepeth colliery it merely burned away because

* *First Report*, page 137. † *Ibid.*, page 108, question 3,151.

it was unconfined; but the members would see that coal-dust was not safe under conditions in which it met with resistance. He would be very glad to afford Dr. Bedson an opportunity of testing some of the exceedingly fine dust which was found about 1,000 yards from the shaft of one of the collieries under his charge. It might be supposed that Dr. Bedson's discussion of the subject was rather above the level of the members' comprehension, but he hoped this would not be considered to be the case, because such a subject must be thoroughly investigated before they could arrive at a proper understanding, and Dr. Bedson had put his views before them in the simplest possible form that the chemistry of the question would permit.

Dr. Bedson said that the gases had so far only been obtained from the coal by heating *in vacuo* at different temperatures. The experiment described, in which the coal was placed in an atmosphere of hydrogen, was performed at comparatively low temperatures. He proposed to extend the investigation so as to embrace the question raised, viz.: whether gases were given off from different coal-dusts at temperatures reasonably within the range of the temperature of the mine. There was no doubt that the physical condition of the coal affected materially the retention by the coal of these gases. Some coals of soft texture would readily yield their enclosed gases, and in the working of such coals considerable volumes of gas would be produced, requiring removal by adequate ventilation. Hard coals containing a larger proportion of inflammable gases might be worked with greater safety owing to their physical structure enabling the coal to retain the gases. With regard to the oxidation, supposing a coal which initially contained hydrocarbon gases was exposed to the air, it would absorb oxygen from the air; and if the temperature rose this oxygen thus brought near to the combustible material would burn it, carbon dioxide would be produced and this would be retained by the coal substance.

Mr. Blackett was afraid that he had not made himself quite clear. What he wished to know was, having accepted the principle laid down by Dr. Bedson, and the coal-dust having been impregnated with carbon dioxide, was it possible for the fire-damp to drive out that carbon dioxide, and take its place?

Dr. Bedson said that if they took the case of coal-dust at Brancepeth, the dust in the body of the hopper contained a very small proportion of combustible gases. Allow that to be exposed to the air, and it would give off the combustible gases and absorb air. The air being absorbed, oxidized the coal substance and carbon dioxide was formed. Mr. Blackett's question involved a knowledge of the relative absorptive power of coal or char-

coal for marsh gas and carbon dioxide, and with our present knowledge it is not possible to state whether marsh gas would displace the carbon dioxide absorbed by coal.

Mr. R. L. WEEKS stated that as a matter of fact the dust at Brancepeth colliery was contained in an enclosed box, and immediately prior to the accident was being shovelled down by two or three workmen. At first it did not explode, and the boys below had to shout up to them to throw no more dust down as they could not see. They then began to throw it down gently and it ignited. If there had been an explosion of any violence, the hopper would have been destroyed, but the only damage effected was the detachment of a sheet of thin iron from the roof, while the thin glass in the windows in the upper portion of the hopper was merely cracked by the heat.

The following paper by Mr. A. G. Charleton on "The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment" was taken as read:—

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part VII.—Medium-coarse to Fine-crushing.

Stamps.

Dr. R. W. Raymond, than whom no one perhaps has contributed more to the scientific literature of practical mining, makes the following remarks with regard to stamps† :—

"In considering the economical application of stamping-machinery, we meet at the beginning with serious difficulties in obtaining accurate information for comparison. The weight and fall of stamps vary as the shoes and dies wear out, and this alone may lead to a change of speed.

"Moreover, defects in engines, boilers, and machinery, for the transmission of power, may occasion serious loss, which cannot fairly be charged to the arrangements of the stamps proper.

"Finally, the hardness and tenacity of the rock crushed varies so much that comparisons between different localities cannot be implicitly trusted. The safest experiments are those made in the same mill by changing first one and then another condition of working, but this is seldom possible for such conditions as weight and lift of stamps, and only within narrow limits for their speed. We may eliminate questions of friction, transmission, and generation of power, in the case of stamps, by measuring the power actually developed by their fall. Thus the weight multiplied into the fall in feet, and the number of drops per minute, gives us exactly the number of foot-pounds exerted by each stamp. Dividing by 33,000 (the number of foot-pounds per minute per horse-power) we have the horse-power per stamp, from which the effective power of the whole mill may be obtained. Dividing the amount of rock crushed daily by the effective horse-power gives us the daily amount per horse-power, and this is the best measure that can be obtained for the effectiveness of stamps.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351; vol. v., page 271; vol. vi., pages 69, 295 and 457.

† *Mineral Resources West of the Rocky Mountains*, 1871.

A complete discussion of the subject would require us to determine the exact influence of the discharge, etc., and the exact resistance offered by different classes of rock, for both of which points the data are wanting."

Prof. Munroe, summing up his paper on the weight, fall, and speed of stamps,* says the following conclusions may be drawn from the data at hand:—

"1. The product of a stamp-battery, other things being equal, is directly proportional to the foot-pounds developed in the fall of the stamp, and to the number of blows per minute, provided only that the discharging capacity of the battery be equal to or greater than its crushing capacity.

"2. Both the force of each blow and the number of blows per minute may apparently be increased per minute up to certain limits with but slight diminution of the rock crushed per horse-power, provided that the discharge be increased in the same proportion.

"3. The discharge of a battery depends on the character of the screen (its area, the size and number of its openings, etc.), and on the number and duration of the splashes produced by the fall of the stamp.† The discharge cannot therefore be increased indefinitely, but up to a certain limit increases in proportion to the speed of the battery, or with the number of drops per minute.

"4. With a limited discharge, equal to or less than the crushing capacity of the battery, the product may be raised by increasing the speed, but cannot be raised by increasing the force of the blow.

"5. A light stamp with a high drop and a heavy stamp with a low drop produce nearly equal crushing effect, provided that the foot-pounds developed are the same. The advantage, however, is in favour of a heavy stamp and a low drop.

"6. To increase the crushing effect of a stamp-battery it is better to increase the weight of the stamps than the height of the drops.

"7. A drop of 8 inches has been generally adopted as giving better results than higher drops. It is possible that still lower drops may prove economical.

* *Trans. Am. Inst. Min. Eng.*, vol. ix., page 84.

† To this writer the author would add—the slope of the screen-plate, ratio of total area of stamp-shoes to total area of mortar, height of screens above the dies, and slope of screen-opening.

"8. The objection to high drops does not seem to apply to high velocity of impact as obtained by spring, atmospheric, or steam stamps, which apparently give results as good as or even better than those from stamps of which the foot-pounds are due principally to weight.

"9. High speed and high velocity of impact lessen the weight of the stamps, and diminish the number of batteries requisite for a given amount of work, and therefore lessen the first cost of the plant. These smaller batteries do not demand large buildings for their accommodation, nor the construction of large and expensive foundations. To offset this saving, must be placed the extra wear and tear due to high speeds.

"10. In the ordinary construction of stamp-batteries with cams and tappets, velocity of impact can be secured only by giving the stamps a high lift: this is incompatible with high speed, but, as we have already shown, under the conditions of limited discharge, speed is more to be desired than high velocity of impact.*

"11. With steam-stamps, atmospheric-stamps, or spring-stamps, both high velocity of impact and high speed can be secured. By the use of machines of this character the maximum effect can be obtained with the smallest plant. The product that can be obtained from one head will be limited by mechanical conditions, and by the impossibility of increasing the discharge beyond a certain limit. The last limitation will operate sooner with fine-screens than with coarse.

"12. In fine stamping both the difficulty of discharge and the work of crushing increase with the fineness of the product. According to Mr. Rittinger the product of the stamp-battery is proportional to $\sqrt[5]{d^2}$, in which d is the diameter of the screen openings. This ratio seems to hold good with the stamp mills of this country [America], though data are wanting to determine this exactly."

The subject has also been ably discussed by Mr. A. N. Rogers, in a paper on "The Mines and Mills of Gilpin County, Colorado."† He

* This has been confirmed by experience at the Caledonia mill, Dakota. It does not hold good, however, with dry stamp batteries where the blow of the stamp both pulverizes the ore and helps to force it out of the mortar. At the Metacom mill, for instance, it was found that with 60 drops per minute, not quite 1 ton per stamp was discharged per 24 hours, 90 drops discharged 2 tons, and 102 drops a little over 3 tons per stamp, the increase in speed thus increasing the yield by 244 per cent.

† *Trans. Am. Inst. Min. Eng.*, vol. xi., page 29.

takes a somewhat opposite view to Prof. Munroe (who supports the Californian system), and argues, from mill practice in Colorado, that light stamps of, say, 500 lbs. dropping 16 inches 30 times per minute are superior in effect to 700 lbs. stamps dropping 10 inches 65 times per minute ;* and he brings forward several practical reasons for his opinion, amongst others that with a low drop and high speed there is great liability for the sands to pack at one end of the mortar, leaving the dies bare at the other, as the agitation in the box gives the ore no time to settle, resulting in loss of work, injury to the mill, and defeat of amalgamation.

In examining the two sides of the question it must be recollected that the duty or amount of stone crushed by any battery, in a given time, depends on the number and weight of the stamps, their height of fall, and number of drops per minute, to which must be added their order of drop, the design (sectional area and position, as well as height of screen-opening) of the mortar, the mesh of the screens, whether hand or mechanical feeding is employed, the quantity of water used, and the regularity or otherwise with which the shoes, dies, and screens wear.

The effect of the more rapid drop in California is to give a less intermittent action, in which it most nearly approaches the idea of a good crushing machine.

It is most important in this connexion that the shoes and dies should wear down flat and evenly ; that the height of drop should be kept constant, and that the bottom of the screen-opening should be regulated to correspond with the wear of the dies, which is effected in Dakota and elsewhere by using a chuck-block. A maximum discharge appears to be obtained by dropping the middle stamp first.

In Colorado, the depth of water, in the deep form of mortar used, exercises an important influence in deadening the blow of the stamp, and weakening the force of the splash, giving the gold time to settle, retaining the pulp in the box, and discharging it in a much finer condition than is the case with a less deep discharge using the same size of mesh screen.

The author's view of the case is, that the choice between light stamps

* Practice has been somewhat modified since Mr. Rogers wrote, both in Colorado and California. In the former locality the rate of drop still remains the same, but the weight of the stamps has been increased to 550 or 600 lbs., and the drop to 18 or 20 inches. In California, on the other hand, the drop has been diminished to between 4 and 6 inches, increasing the weight of the stamps to 750 or 850 lbs., and running them faster, at from 90 to 105 drops per minute. In some cases (as at the Grass Valley mill) the drop runs up to 7 inches, and the stamps weigh occasionally as high as 900 to 1,000 lbs. with lower drop.

with high drop and low speed, and heavy stamps with low drop and high speed can only be settled by knowing the character of the ore, whether it can best be crushed fine or coarse, and deciding on the nature of its treatment according as a large output or close saving of free-gold is the vital point to be considered.

The Colorado practice has been developed and proved correct, using a deep roomy mortar for fine-crushing, and amalgamating mostly in the battery; while the Californian practice, which aims at crushing coarse and catching a large proportion of the gold outside the boxes has equally strong, if not stronger, features to recommend it (with certain grades of ore) on the Pacific slope.

The question, in fact, to be determined first of all, is whether the stamps themselves are to be used as a close-saving amalgamating-machine, as well as for crushing, or whether the latter object is the one to be kept most in view—trusting to extraneous gold-saving appliances, to catch whatever gold escapes the stamp-mortar.

Whilst at the present time the theoretical work done by California and Colorado stamps is about the same, for the reasons that have been stated the actual work done does not correspond in the two cases. In Colorado the stamp crushes 1 ton per 24 hours, while in California with stone of similar hardness the amount varies from two-and-a-half to three times as much.

The relative wear and tear of iron is shown by Mr. Rogers (from examples cited) to be greatest in the Californian mills, amounting to 1·8 lbs. worn from the shoes per ton of rock crushed, and 0·9 lb. worn from the dies, compared with 1·44 lbs. loss of shoe-metal, and 0·31 lb. of die-metal in Colorado; assuming the ore reduced in both cases to the same standard of fineness.

As the Californian battery-pulp is much the coarser, in reality, the figures in actual practice are however reversed, 70·8 tons being crushed in Colorado per shoe, and 78 tons per die, before removal, whilst 79 tons per shoe, and 100 tons per die, are crushed in California. The proportionate breakages are 1 stem per 25,000 tons stamped in Colorado, and 1 stem per 864 tons in California.

Mr. Rogers argues, from the foregoing data, that the destruction of metal per unit of work will be inversely as the efficiency of the battery, regarding the energy stored up in the stamp, as a force, which, if not expended on the rock, will occasion an abnormal wear of the surrounding metal. On this assumption, from various figures given, he deduces: first, that the Grass Valley mill (taken as an example of Californian practice)

wears out most metal per ton of rock crushed, and has the greatest proportion of wear on the die; second, that the Bobtail (Colorado) mill performs one and one-half times as much work for each pound of metal worn from the die, and twenty-three times the work for the breakage of a stem as the other mill; third, that the shoe of the Bobtail mill wears four and six-tenths times as fast as the die, thus proving that the blow is taken by the rock, and not passed through the stone to the die beneath; fourth, that while the Bobtail mill wears the shoe four and six-tenths times as fast as the die, the other mill wears its shoe but twice as fast as the die, which indicates that more of the work passes through the rock into the die, employing the same as an anvil; fifth, that the relative endurance of the stems in the two cases must be taken as the most conclusive evidence that the blow of the Bobtail stamp, notwithstanding the velocity of the impact due to greater drop, has been absorbed in the rock, and the stem has not received the violent shock, which would result from falling upon the metal of the die; and sixth, that by careful analysis of these data no undue wear can be detected in the battery or foundations of a high drop mill as compared with a low drop one, and therefore its work must have been properly expended upon the rock in the battery.

Theoretically, these arguments are interesting and plausible, but it strikes the author, that to give them practical weight, for relative comparison between the two different systems, we ought to know the relative hardness of the rock taken for the test, the relative quality of the metal used for shoes, dies and stems, and the construction of the battery-foundations in each case. The relative endurance of the stems alluded to by Mr. Rogers in his fifth contention, may perhaps be accounted for, by the different taper or diameter of the end (just above which, the stem generally breaks) or to want of attention in annealing the stem after welding.*

To express a fair opinion, it is also necessary to know the position of the guides, and the relative attention paid to them, as if they are allowed to wear or become loose, undue breakages will result. Moreover, the not less important question of feed must be considered, for if the stone banks-up in one end of the box, leaving the dies bare at the other, or is overfed, it will tend to a similar result, though the author admits that there is less probability of this happening with a high drop, than with a low one. In other words, less attention is required in feeding in the former case.

* To change a stem, it saves much time and trouble if a differential pulley be used, attached to an overhead crane running on rails the whole length of the batteries, at a sufficient height above them.

The slow drop and turn of the Colorado stamp makes the shoes and dies wear less evenly than in California stamps; the interval between each drop in the former case being 2 seconds, whilst in Californian batteries it varies from $\frac{3}{5}$ to $\frac{2}{3}$ second.

Regarding the Colorado and Californian systems, without any prejudice either way, there is another point which the author thinks should not be overlooked, viz., the previous and subsequent treatment of the pyritic portion of the ore which contains gold.

Where a certain portion of the gold-ores of a district are of a sufficiently massive character (as they are in Colorado) to be capable of hand-selection, it frees the battery of a certain amount of work, and on this account capacity is not so much an object as it is when every particle of payable stone requires to be stamped. The portion of the ore which thus escapes the battery treatment contains the bulk of the heavier pyrites, and its removal therefore indirectly influences the results of concentration, and the after disposal of the concentrates upon which the manipulation of the ore directly depends. With fine-stamping for instance, such as obtains in Colorado, owing to the free-gold present being very fine and intimately associated with the pyrites, one would expect to find the losses in pyritically-combined gold, greater in concentrating them on end-bump table or blankets and buddles, than with the coarser stamped Californian ores, for which mechanical belt-concentrators are almost exclusively used.

These losses are neutralized, however, in Colorado practice, by the removal of the heavier portions of the ore (containing much of the pyrites) before it is subjected to the stamp-process, and the Colorado pyrites being generally less valuable than the Californian the losses (chiefly of silver) in concentration are less felt and apparent. Further, while the hand-picked pyrites and fine concentrates are well adapted to the smelting treatment,* which obtains in Colorado; they† would be far less, if at all fitted, for the chlorination method, which holds its own on the Pacific slope, as the selected ore would have to be all crushed dry with stamps, or rolls, and the fine concentrates would tend to clog the filters.

* The low tenor of the concentrates in gold, and the proportion of silver they contain, coupled with a preponderance of copper pyrites and grey copper, and the presence of arsenic in the ore, with abundant fluxes close at hand, are conditions which render a matte-smelting far better adapted to these ores than any other, whilst the chlorination process is most suitable, on the other hand, for the siliceous Californian concentrates, in which iron pyrites predominates.

† The author does not of course here refer to ores (carrying considerable quantities of lead), which would be quite unsuited to chlorination, but only to pyritic ores, comparable with those of California.

The gangue of the Gilpin county mines in Colorado is more feldspathic than quartzose, being the product of the alteration of the granitoidal-gneiss and quartz-andesite dykes which penetrates it.

The mill-stuff treated in Amador. Calaveras, and Tolumne counties (California) on the other hand, is of a different description, essentially quartzose, carrying 1 to 2 per cent. of pyrites, but mixed with a large percentage of country-rock, which in this case is slate, augite-schist and diabase, the slate predominating, and the gold being coarser and less intimately associated with the pyrites.

It may be noticed, that the proportion of smelting to free-milling ore, raised in Gilpin County, Colorado, during the year ending May 31st, 1880, was as 10,218 to 113,427 tons, the selected stone having an average value of £10 1s. 6d., as compared with £2 1s. 3½d., that of the mill-rock.

According to Prof. Egleston,* the distinguishing feature of the auriferous smelting-ores of Colorado, is the presence of manganese,† which, while it is oxide on the surface, produces below the water-level beautiful crystals of carbonate, and also by the presence of copper (carrying both silver and gold) instead of lead,‡ as is usually the case elsewhere. The relative fineness of crushing in the Colorado and Grass Valley mills, is

* *Metallurgy of Silver, Gold, and Mercury in the United States*, page 406.

† Manganese often forms part of the gangue of gold and silver veins, and when not superabundant may be considered a favourable association for the precious metals.

‡ By far the greater portion of the silver ores of Colorado are reduced by silver-lead smelting in water-jacket furnaces. This is due to the metallurgical skill shown in decreasing the loss in smelting, by lowering the percentage of lead required, and by the unusual economy with which the process is carried on. Of late there has been a larger use of calcining-furnaces for the preliminary treatment of refractory ores to be smelted. The ores are not reduced at the mines, but are shipped either as crude mine ore or are concentrated at the point of reduction, and the concentrates sent to customs smelting works. Even when the ores are dry (that is containing at most a low percentage of lead) they are usually sent to the smelters for admixture with other ores, with better metallurgical and commercial success than if treated by amalgamation or lixiviation. This practice is largely due to the usual failure of attempts to operate silver-lead furnaces on the ores of single mines or districts, and to the practice of selling ores and concentrates through sampling works which partly conduct a commission and brokerage business by correctly ascertaining the value of the consignments, and selling the lots to the highest bidder. This system has been specially advantageous to the smaller mines, as it obviates the cost of erecting reduction works and affords immediate cash returns for the producer. The smelting plants established at central points, such as Denver, Leadville, Durango, and Pueblo, are thus enabled to obtain as nearly as possible a typical smelting mixture at all times, which can be worked to advantage, and large quantities of ore can be treated which could not be profitably smelted alone. The chief difficulty with this system is the cost of freight when the ore has to be brought from outlying districts, but this diminishes as the country is opened by roads, and railways.

stated to have been as 80 to 40 mesh, only 1·17 per cent. of the Bobtail ore being caught on a 40 mesh-screen, while the fineness of the tailings of the two mills was as 3 to 2.*

Mr. Rogers says, that the saving by amalgamation above the blankets was fully 70 per cent. of the contained value of the gold in the ore, and about 6 per cent. of the silver, and the cost of milling but little more than 4s. 2d. per ton,† embracing current expenses, repairs, and renewals.

The average expense in Colorado, in the early days, was stated (the writer believes, by Dr. Raymond) as :—

	£	s.	d.
Crushing and amalgamating mill-rock	0	16	0
Treatment in pans and dolly-tub ...	0	0	7
Concentrating and hauling	0	0	11
Extracting gold from concentrates ...	0	3	0
Total per ton	1	0	6

The summary of operations for 1878,‡ shows that 22,936 tons were crushed at the Bobtail mill, out of which 75·80 per cent. of the gold was saved by direct amalgamation, and 58·96 of the mineral was caught in the concentrates. The loss of mercury has been stated by some authorities as being largest in the Colorado practice, but Mr. Rogers combats this idea, alleging that in well-managed batteries, it only amounts to 0·02 pound per ton of ore.

The width and roominess of the Colorado mortar present conditions favourable for allowing the pulverized ore to escape the falling stamps, and assisted by the deep discharge, retains the finely crushed pyrites inside, whilst the long drop gives the interval of time necessary for the gold to settle on the plates in the box where it is amalgamated.

Mr. Rickard considers the results of an interchange of treatment using Californian batteries on Colorado ore, or *vice versa*, would be as follows :— The Gilpin county ore is worth, say 8 dwts. or 8 dollars per ton. The local methods extract 5·60 dollars by amalgamation, without counting subsequent concentration, at a cost of 70 cents. A Californian mill crushing over $2\frac{1}{2}$ tons of ore, at a cost of 25 cents per ton in California, would give an extraction in Colorado of only 4·00 dollars, but would crush the Colorado ore three times as fast as a Colorado battery, so the cost would be, say 25 cents for a net yield of 3·75 dollars as against 4·90 dollars obtained by present methods. Using the Gilpin

* According to Mr. T. A. Rickard, it has been proved by actual test, using a 40 mesh-screen, that fully 70 per cent. of the pulp in Colorado and 50 per cent. in California will pass a 100 mesh-sieve.

† Table II. shows that the cost at the present day is still less in some mills, as for instance the Hidden Treasure, where it only costs 3s. 3d. per ton.

‡ Statement by Mr. G. W. Gray.

county mill in California, treating an ore containing 6 dwts. and worth 6·00 dollars per ton, the Californian mill would extract 70 per cent. at a cost of 35 cents, leaving a balance of 3·95 dollars per ton. The Colorado mill would extract an increased percentage, say 75 per cent., but the ore being harder it would crush less, and hence the cost per ton would be greater than in Colorado, say 1·00 dollars per ton, leaving a net yield of 3·50 dollars per ton. The Californian mill would, if crushing 100 tons a day, therefore show a profit of 45 dollars per day more than the Colorado battery. Other factors that have influenced the choice of methods have been the smaller size of the Colorado ore-bodies as compared with those of California, whilst the construction of a Colorado mill of a capacity equal to a Californian plant, would require double the capital outlay.

Outside of Colorado and Californian methods, there is a third system of milling. The author refers to the Dakota practice* of employing 850 lbs. stamps, dropping 9 to 12 inches, running at 74 to 85 blows per minute, and using a mortar-box, which possesses peculiar features of its own, whilst it meets, to a certain extent, the Colorado requirement of depth. The principle to be observed is to feed low; the feeders seeing that the height of the ore, between the shoes and the dies does not exceed an inch, and as much less as possible, without the shoes commencing to pound. The screens used are 30 mesh wire, or corresponding slot-punched.

The loss of iron from wear of shoes is 0·35 to 0·37 lbs., and from dies 0·40 to 0·48 lbs. per ton, using chilled white-iron shoes, and grey mottled iron (chilled at the top) for the dies. The loss of quicksilver is usually 0·0011 to 0·0044 lb. per ton of ore crushed, and not more than 5 stems are required yearly for a 60 stamp-battery. A careful record of all wearing parts of the machinery about a mill and their individual life, ought to be kept by the management, if operations are to be conducted in a business-like manner.

With close-fitting wooden tops to the boxes, catch-pans, under-bearings, and care in the use of grease in places where it might drop into the mortar, there ought not to be much trouble from that source. The cams, under ordinary circumstances, should be lubricated with axle-grease, and a curtain spread below them, to catch any grease that they may throw off whilst in motion.

The groves in the guides are lubricated with a preparation of black-lead and linseed oil, mixed warm, in such proportions as to form a stiff

* With an ore containing any considerable quantity of pyrites (the same as in California) care would have to be taken not to overcrowd the concentrators in adopting this method.

paste. Oak guides last eighteen months, and pine only two. Failing the use of Broughall or Fargo sectional-guides, instead of grooving the ordinary guide-blocks to receive the stem, a good practice is to cut a dove-tailed recess opposite each stamp, in which a wooden concave-faced key can be fitted, so that the grain of the wood is parallel with the stem instead of across it.

In wet crushing 1 to 1¼ tons of ore is an average amount to crush per horse-power. The amount of fuel required to run the stamps, depending on their weight and the load of the other machinery, averages 0·22 to 0·25 cord of wood per stamp.

A novel feature in the Homestake practice has been the introduction of screens of aluminium-bronze, which are stated to have proved very satisfactory as a substitute for Russian-iron. The bronze, supplied by the Cowles Electric Smelting and Aluminium Company contains 5 per cent. of aluminium, 95 per cent. of copper, and a trace of silicon, and is furnished in unperforated sheets at 1s. 10¼d. per lb. The sheet is 0·038 inch thick, lasts six months, and does not break or get unserviceable through wear, while the life of an ordinary Russian-iron screen seldom exceeds two weeks, and is sometimes much less. This may be an item of economy, small as it may seem, worth noticing.

The choice of any special make of automatic-feeder, it has been shown, chiefly rests on whether the ores are dry, or damp and sticky, as some classes of feeder suitable for the one, are not so for the other.

As the mills of Dakota have to treat low grade ores, it is necessary for their profitable operation that large quantities should be dealt with as rapidly as possible, saving as much gold as practicable by the simplest means. Consequently, the mills in the Black Hills amalgamate both inside and outside the mortar in order to avoid the more expensive auxiliary operations necessary to recover the gold used in Californian mills. It is evident that the Dakota practice differs from that of both Colorado and California in that it permits of the large capacity of the Californian mill, and at the same time of adopting the simple plan of amalgamating inside the box to advantage (as is done in Colorado), hence it is a sort of compromise between the two systems. The discharge of the Californian mortar being so shallow (only about 4 inches), inside plates are not admissible, the constant and violent agitation scouring the amalgam off them and preventing the gold settling.

The proposition of employing light stamps with a small area of shoe to mitigate dead-stamping, is a suggestion the author does not agree with, and, in fact, it has been shown that the proposed cure is calculated to

aggravate the disease. To escape dead-stamping, the point is to disentangle the particles of gold, as far as possible at one blow, without the head requiring to drop a second time, and to at once get the pulp out of the mortar-box. This is most likely to happen with a head of heavy weight and large area, providing the battery openings are made of sufficient size, and screens of proper mesh are employed, for facilitating the ready escape of the ore. With these precautions, a low discharge, and a sufficient supply of water to clear the boxes without flooding the plates, dead-stamping need never happen, if the stamps are speeded right, and dropped in proper order, with sufficient lift; at any rate, what little there is, ought to be regarded as a lesser evil, than reducing the duty of the battery by diminishing the weight and area of the head. Dead-stamping is, of course, more liable to occur if the gold in the ore be coarse.

Dead-stamping is least liable to happen with ore like that of Clunes, where the gold occurs in seams and cavities in more or less honeycombed quartz. With such an ore Mr. Rickard points out that a certain blow will break the brittle quartz and liberate the ductile gold. But such ideal conditions are rare, and more frequently the stamp has to deal with material of such uneven size as regards the metal and the gangue that the particles are apt to become confused together, and the one is often crushed too much and the other too little. Owing to the greater friability of the sulphide minerals, the latter are generally reduced to much greater fineness than the stone, and hence if the gold is not very intimately associated with the pyrites, or in too fine a condition, coarse stamping is often sufficient to liberate it.

On the Continent, where rectangular non-revolving heads* are employed, the tendency of late years, at Clausthal for instance, has been to set the heads as close together as possible, to give the greatest possible crushing-surface between shoes and dies. The Caledonia mill in Dakota, using stamps 9 inches in diameter, weighing 850 lbs., and dropping 85 times per minute, claims to save 85 per cent. of the free-gold in the stone (60 per cent. of which is caught on the inside plates); and that there is but little dead-stamping, is vouched for by the fact that the batteries put through no less than $4\frac{1}{2}$ tons of quartz, mixed with mica and amphibole-schist, per head

* The rotatory action of the stamp (making with high-speeded batteries a whole revolution in 9 to 10 drops, and with slow stamps one complete turn for each fall), whilst it may tend to abrade the surface of the gold and rub off any film of foreign matter likely to interfere with amalgamation, produces the bad effect of tending to slime any sulphides present, which accounts for the preference in Germany for non-revolving heads; stamps being used there mostly as an auxiliary crushing-machine, for after-concentration of the ore.

per diem. Mr. Melville Atwood has drawn attention in an article, entitled "The Microscope in Metallurgy,"* to the deleterious effect which even a small quantity of zinc (blende), if present in the ore, produces in battery-amalgamation through sickening the mercury. He also states that when a large proportion of pyrites is present in the ore of the Bodie mine he found that the battery-water had a decidedly acid reaction, due to the presence of soluble ferric and ferrous salts. The zinc destroys the action of the quicksilver, enfilming it with a compound insoluble in water, preventing the metallic contact so necessary to amalgamation taking place between the detached mercurial globules. The same happens if arsenical pyrites be present.

In experimenting on some of the Wynaad ores, the author found great difficulty in keeping the copper-plates free from yellow-stain (until thoroughly coated with gold-amalgam), and he attributes that circumstance to the softness of the water, causing it to absorb a large amount of air, which even more, perhaps, than carbon dioxide tends to form ferric sulphate (the most active agent in staining the plates) when in contact with the pyrites. As a remedy in similar cases, a small quantity of caustic lime might be introduced into the water-supply tanks. The water referred to had a hardness of 2 degs. by the Clark test, and contained 6·5 grains per gallon of solid matter consisting of carbonate and sulphate of calcium and a trace of silica. Air acts prejudicially in another way, as it is naturally absorbed in the water by the violent action of the stamps in the form of bubbles; and whilst the quartz is broken into irregular grains, owing to the more highly-developed cleavage of the metallic sulphides, they are broken up into thin grains and flakes, which the air has a tendency to carry off in suspension. Artificially raising the temperature of the water, and so causing the air to expand and the bubbles to burst, is undoubtedly the only means of overcoming this difficulty.

In connexion with copper-plates, the outside aprons ought certainly not to be cleaned oftener than absolutely necessary. The gold-amalgam should, in fact, be allowed to settle on them, forming ridges which present surfaces for other particles of gold and amalgam in suspension in the water to catch upon and cling to. The amalgam must not be allowed, however, to become too hard, or pieces of it are liable to break off and be swept away. Should this happen, it must be at once rectified by throwing a thimbleful or two of mercury inside the battery, or sprinkling a little over the copper-plate. Nearly every millman has his own ideas about the

* *California Mining and Scientific Press*, June 24th, 1882.

arrangement, preparation, and care of copper-plates, and such like matters. Loss is liable to occur by giving the tables too much or too slight a gradient,* and by using an excess or an insufficiency of water, both of which points must be carefully attended to. In some mills, as elsewhere stated, silvered copper-plates and rocking-tables are used (with a riffle attached) for catching truant amalgam. It is a good plan to put a perforated pipe over the ordinary riffles, directing a number of water-jets on to them to prevent clogging, if, as sometimes happens, the turning on of more water into the box itself would make the pulp too thin, preventing proper settlement of the gold, and discharging the ore in too coarse a condition. So long as there is no actual movement of the tables longitudinally or laterally, the general vibration caused by the rise and fall of the stamps produces a kind of pulsation in the water flowing over them, assists gravity, and is favourable to the settlement of the gold and amalgam. The advantages of stamps, to the writer's mind, consist in their simplicity of mechanism, durability, and large crushing capacity, coupled with the fact that if one or more heads get out of order it does not throw the whole mill out of gear; nothing being more fatal to economy than a whole plant standing still, since every minute lost, multiplied into the total staff employed, soon mounts up to a large figure in pounds, shillings, and pence.

Then, again, the repairs are light, and can be mostly done by a good blacksmith; the capacity of the mill can be regulated to constant variations in the supply of ore by starting or hanging-up a battery (5 heads); and, finally, stamps extract a large percentage of gold in those cases where it is free, without any accessory machinery.

The preliminary step towards banking gold, is like jugging a hare, to first catch it with as little delay as possible, and that is what is done in the mortar of the stamp-battery. The man who christened the South African gold formation *banket*, perhaps took a somewhat sanguine view of the case, if he expected to do without the intervention of even the simple stamp, as a means to that desired end; but he evidently had the right principle in his mind's eye.

So long as 70 to 85 per cent. of the free-gold contents of the stone can be extracted with stamps, independent of all other appliances but copper-plates inside and outside the boxes, and a few mercury-traps, the battery (a good old conservative piece of mechanism) will remain a formidable

* The heavier the ore the greater the gradient required. The inclination most effective for settling the gold may, however, cause the pyrites to settle, and too heavy a gradient may sweep both away.

rival to many so-called improved processes, for which a liberal royalty has not unfrequently to be paid, for the doubtful privilege of experimenting with them. Its simplicity and capacity explains the extreme cheapness of the process in such instances as the Homestake group of mines, the Alaska Treadwell, and the Morgan mine in Wales.

For the figures the writer has been able to give in table II. part IV. of this paper,* in regard to the last-named property, he is indebted to Prof. C. Le Neve Foster, Mr. N. T. Williams, and the makers of the machinery, and he is pleased to be able to state, that Wales, as represented by the Morgan mine, under Mr. Williams' management, can claim to have crushed gold-quartz more cheaply than any stamp-mill in the world.

The expense of producing gold at the mines of the Golden Leaf Co., at Empire in Montana, may also be specially cited as an instance of cheap mining and milling. The cost of mining and milling per ton is given by Mr. H. M. Beadle as follows:—

	Empire.		Bell Boy.
	1891.	1892.	1892.
	s. d.	s. d.	s. d.
Mining	4 2	4 $2\frac{1}{2}$	7 $10\frac{1}{2}$
Mine development	2 0	1 5	1 $5\frac{1}{2}$
Milling	3 $3\frac{3}{4}$	3 $3\frac{1}{2}$	3 2
General expenses (haulage included)	0 $10\frac{1}{2}$	1 $10\frac{1}{2}$	5 $3\frac{1}{2}$
	10 $4\frac{1}{4}$	10 $9\frac{1}{2}$	17 $9\frac{1}{2}$
Quantity mined and milled ... Tons	53,700	46,600	10,880

For the sake of comparison, the cost of crushing in some of the best known California stamp-mills is given below:—

Name of Mill.	No. of Stamps.	Cost per Ton.
		s. d.
Moore	10	3 4
Stickles...	20	1 8
Sheep Ranch	30	4 2
Idaho	35	10 0
Bunker's Hill	40	2 6
Keystone	40	3 $1\frac{1}{2}$
Plumas Eureka	60	2 0
Plymouth Consolidated ...	160	1 $7\frac{1}{2}$

In the three instances specially alluded to, amalgamation is carried on in the battery, and no extraneous appliances are used, but copper-plates, riffles, and mercury-traps.

Stamps certainly do possess some defects however. They are not suited for instance for an ore (*a*) which contains pyrites in large quantity, which cannot be sorted-out beforehand, (*b*) which requires to be pulverized very coarse, (*c*) or which requires for its after-treatment to be gradually reduced in several successive stages to a certain size; for such work, rolls have a decided superiority. For direct and continuous

* *Trans. Fed. Inst.*, vol. vi., page 76.

operation, grading the material closely, pulverizing finely, evenly, and cheaply, discharging the waste as soon as disengaged (thus freeing the amalgamation of a hindrance, and the mill of an encumbrance), as a well-known authority has remarked, stamps have, however, never yet been matched, whilst they take advantage of the brittleness of the rock in crushing it.

Mills.

The Huntington, Schranz, and Tustin mills, and edge runners, like the improved Chilian mill, as well as Frankfort mills, have some advantages over stamps, as their tendency is to rub and brighten the gold, which renders it more easily attacked by quicksilver.

Considering the results it is reported to have achieved, the Schranz mill seems to deserve to be better known than it is, outside of Germany. Its essential parts are a large slightly coned ring or annular plate, revolving about a vertical axis at the rate of $12\frac{1}{2}$ revolutions per minute, and three conical rollers, with fixed inclined axles, which are radial with respect to the central vertical axis of the machine. The rollers are set on the plate 120 degs. apart, and rotate by frictional contact. An illustration of the machine is given in Mr. Linkenbach's *Aufbereitung*. It claims to minimize the production of fine meal and pulp, requires $3\frac{1}{2}$ horse-power to run it, and working on 0·12 to 0·32 inch (3 to 8 millimetres) quartzose jig-tailings, has been proved to have a capacity of 3,200 lbs. per hour, with a consumption of 31 gallons of water per minute ($5\frac{1}{4}$ gallons out of this amount being used for the sizing-drum). The durability of the wearing parts (the crushing-surfaces being of the toughest Bessemer steel) is a special feature. For ore not exceeding $\frac{1}{3}$ inch in size, it seems as if this mill ought to take the place of stamps in some concentrating works, as experiments in reducing ore of 5 and 8 millimetres (0·197 and 0·315 inch) size down to 2·4 millimetres (0·01 inch) both by stamps and by mill, gave the following interesting results:—

Proportional Quantities of Ore of Different Sizes produced in Crushing, expressed in Percentages.

Millimetres Inch ..	3·2–2·4 0·126–0·094	2·4–1·6 0·094–0·063	1·6–0·9 0·063–0·035	0·9–0·5 0·035–0·020	0·5–0·2 0·020–0·008	Less than 0·2 Less than 0·008	Second Pulp Settlings.	Finest Basin Settlings.
Stamps...	—	4·68	15·15	16·96	24·08	16·72	3·88	12·66
Schranz mill ..	6·95	21·07	26·27	16·92	15·31	7·21	1·71	1·83

This table shows that in using the Schranz mill, 64·26 per cent. of the material was obtained in a settling run, between sizes 2·4 and 0·5 millimetres (0·094 and 0·020 inch); 15·31 per cent. consisted of fine

meal and coarse pulp, and 12·98 per cent. was fine pulp; while the corresponding figures for coarse stamping were 36·78, 24·08, and 39·13 per cent.* The writer may also observe that Frankfort and Chilian mills are both an exceedingly good form of combined comminutor, agitator and amalgamator, more so, he thinks, than pans for gold-ores, and in proof of it, he need only point to the good work the former are doing in Italy (mentioned by Prof. C. Le Neve Foster) and his own observation of the latter in Mysore. In fact, if gold sulphides must be ground and amalgamated raw, he considers a Chilian mill one of the best machines to employ for that purpose.

The chief case in which the Huntington mill distances other competitors is in the treatment of comparatively soft clayey ores, which neither a stamp-batttery nor rolls are well adapted to handle; whilst conversely, of course, the Huntington mill is not as well fitted as stamps for crushing very hard rock.

The writer thinks that the stamp-battery is *par excellence* the machine for most brittle ores, just as rolls, and, in exceptional cases steam-stamps, are for tough ones, but he would give the preference, as he has remarked, to the Huntington mill for soft clayey ores, especially such as contain rusty-gold. The portability of this crusher, the comparative cheapness with which it can be set up and used, just like a stamp battery, without mixing, grinding, or other auxiliary plant, or elaborate superstructure, must specially recommend it in those cases where it is desired to give a free-milling ore a fair trial in an out-of-the-way place, at a minimum expense. In evidence of its economy the author may cite the record of the Spanish mine, California, mentioned by Prof. Egleston,† as it is the cheapest instance of gold-ore reduction on record, costing per ton :—

	s.	d.
Labour of tending and feeding, cleaning-up, and superintendence	0	4·50
Water for mills and rock-breaker	0	1·80
Shoes	0	1·45
Screens	0	0·65
Dies	0	0·85
Caps, scrapers, and bolts	0	0·10
Renewals of working parts	0	1·00
Mercury, at £8 6s. 8d. per bottle	0	0·25
Oil (illuminating and lubricating) for mills and breaker	0	0·10
Depreciation	0	0·50
Cost of milling per ton	0	11·20
Labour at breaker	0	1·00
Wear and tear at breaker	0	0·25
Total cost per ton	1	0·45

* Mr. Kunhardt "Ore-dressing in Europe," *School of Mines Quarterly*, series 2 (New York).

† *Metallurgy of Silver, Gold, and Mercury, in the United States*, vol. ii., page 419.

The gangue of the Spanish mine is a decomposed slate, carrying stringers of quartz, with clay-partings between the slate and the quartz, and the ore is very poor, only averaging about 2s. 6d. to the ton.

During the last four months of 1886, the four mills employed at these works crushed 136 tons per day. The mill is driven by an 8 feet Pelton wheel, with 15 inches of water and a pressure of 320 feet, at 60 revolutions per minute; 22 horse-power being required to run the plant, which consists of a Blake crusher, Challenge feeders, and three 5 feet and one 4 feet Huntington mill. Seven shoes a month are worn out by the four mills. The screens are No. 5 slot, and last only four days. The 5 feet mills grind 35 to 40 tons a day each,* and the smaller one 30 tons. Mercury is added to the mills in the proportion of 1 ounce to 4 tons of ore, and 1 ounce is lost for every 15 to 30 tons of ore crushed, the exact amount depending on its richness.

The entire plant cost as follows :—

	£	s.	d.
Four Huntington mills and self-feeders	1,322	2	7
Labour in erecting and building	480	3	5½
Silver-plated amalgamating-plates	413	12	3½
Waterpipe and wheel-shafting and pulleys ...	254	6	5½
Lumber of building and flume	248	17	2
Hardware...	213	15	4
Blake crusher	128	16	10½
Cost of milling-plant under cover and running	3,061	14	2

Freight cost £5 a ton from San Francisco, lumber cost 9s. 4½d. per 100 feet.

The total cost of mining and milling, estimated on a production and reduction of 4,047 tons in 29 to 30 working days, during November, 1887, was 2s. 2d., which gave a profit of 6½d. per ton (on mining and milling) at the Spanish mine : a result which probably eclipses any that has ever been obtained from a low-grade gold-ore. 45 per cent. of the gold caught is saved round and inside the mills, and 55 per cent. on the copper-plates outside ; very little gold amalgam it is said is flowered.

Mr. T. B. Morse expresses the opinion with reference to Tustin mills, which he states have worked satisfactorily at the Willard Mine, Calaveras county, California (where they have been used for crushing ore for amalgamation on copper-plates, and concentration of the sulphides), that a comparison of the two methods of crushing with stamps and mills, shows that a much larger proportion of slimes are produced by the former. As a consequence, he found in trials made with a very hard ore, in which the gold was exceedingly fine, and the sulphides soft and brittle, that a much greater percentage of fine gold was amalgamated after the mills than after

* This seems above the average, which is usually reckoned at 20 tons with a 5 feet mill.

the stamps. He put the saving in concentration after stamping at only 18 to 20 per cent. of the assay value of the ore, whilst with mills he claims to have saved 85 per cent. With the same ore and with the same screen he considered that one mill was equivalent to 6 or 8 stamps.*

Granulators.

Direct crushing in one process has been effected by two very ingenious machines, the Sturtevant mill, and the Gates granulator. The former consists of a chamber of cast-iron fitted with perforated screen plates having oblong openings about $\frac{1}{4}$ inch wide, fitted on the top with a hopper for feeding ore. From opposite sides two cups or cones of chilled iron enter the chamber through openings provided with exchangeable collars; and the cones revolve in opposite directions at about 900 revolutions per minute. The ore, which must be absolutely dry, is charged in lumps of probably about 4 cubic inches; and when the chamber is filled the ore packs itself against the revolving cones, and the grinding action commences. The ore is rapidly crushed and passes out through the slots in the screen-blocks. The dust is drawn off by an exhaust fan connected with the hopper under the machine. The product is screened and the returns go back to the mill to be reground. The capacity of the machine with a suitable material that does not require to be crushed too fine is enormous. A mill with 20 inches cups or cones will (it is stated) crush about 20 tons per hour of calcined ore to 12 mesh, at a cost of 11d. per ton. The Wear amounts to only about $\frac{1}{4}$d. per ton crushed, but on hard ore the wear is very considerable, and rolls in such cases are found more economical. The mill is said to require about 49 to 115 horse-power, according to the hardness of the ore, on the other hand it simplifies the plant and reduces the first outlay. The Gates granulator is the same in principle as the Gates crusher, but the segments composing the outside hollow cone are supported by springs yielding under a certain pressure. Samples of ore crushed by this machine show a very superior and even granulation.

Mr. Sahlin and Mr. S. R. Krom† give a number of interesting figures by which the work of the Sturtevant mill as compared with crushers and rolls may be gauged. Mr. Sahlin, experimenting with Port Henry iron-ore crushed by the Sturtevant mill, passed it over the identical sizing-screens used for the ore granulated by crushers and rolls, with the following results:—

* *Report of the California State Bureau*, 1886.

† *Trans. Am. Inst. Min. Eng.*, vol. xxi., page 534.

Size of Particles. Mesh.	Crushers and Rolls. Pounds.	Per Cent.	Sturtevant Mill. Pounds.	Per Cent.
10–16	310	31·0	... 22½	22·50
16–24	210	21·0	... 21¼	21·25
24–40	79	7·9	... 9½	9·25
40–60	175	17·5	... 29½	29·50
Above 60 ...	209	20·9	... 15½	15·50
Loss	17	1·7	... 2	2·00
Totals ...	1,000	100	100	100·00

This test appears to decide in favour of crushers and rolls, but it is not a fair one, the conditions being altogether favourable to the Sturtevant mill. Mr. Sahlin states that the fine dust had previously been all drawn off by the exhaust fan in the Port Henry sample. The ore as received was absolutely clean and free from dust, whilst it is said that the rapid accumulation of worthless dust in the collecting room is a serious source of loss and embarrassment. Moreover, the Port Henry ore crushed in the Sturtevant mill was a clean, clearly defined, coarse crystalline magnetite, containing only a small percentage of dust-producing minerals, while the heterogeneous mixture treated in the above experiment by the rolls did not carry over 30 to 38 per cent. of iron, and contained besides a large amount of dust-producing materials, such as felspar, olivine, and apatite, all of which remained in the sample when sized. Another point in favour of the crushers and rolls was the smaller amount of power required for a certain hourly production. To illustrate this, a 20 inches Sturtevant mill (stated to have a capacity of 20 tons per hour), with requisite auxiliary machinery, is compared in the tables below with a crusher and roll plant of similar capacity. It is assumed that both plants reduce the ore to 10 mesh size.

Crusher and Roll Plant.

	Horse-power.
Coarse-crusher (24 inches by 14 inches)	20
Two fine-crushers (5 inches by 20 inches) ...	20
Three pairs of rolls	45
Elevators and conveyors	10
Loss in transmission (30 per cent.)	29
Total	124

Sturtevant Mill.

	Horse-power.
Coarse-crusher (20 inches by 14 inches)	20
Sturtevant mill (20 inches)	110
One pair of rolls	15
Elevators and conveyors	10
Loss in transmission (30 per cent.)	46
Total	201

This shows a saving of 41 per cent. in favour of crushers and rolls; and in a day's run of 20 hours, 77 horse-power saved, represents a saving of 6·93 dollars or about 1¾ cents per ton of ore crushed, calculating 3 lbs. of coal per horse-power per hour and coal at 3 dollars per ton.

As to repairs, it is more difficult to find a basis of comparison between the two systems. At the Port Henry mill, it is said that the wear on the cups or cones amounted to one new set per 600 tons of ore ground. These cups are said to weigh about 800 lbs., and their cost can certainly not be less than 3 cents per lb., or 24 dollars per set, which is equal to 4 cents per ton for grinding-cups alone, irrespective of cost of screen-blocks, collars, bearings, etc. The life on hard ore of a pair of 26 inches steel tyres, costing about 200 dollars, averages, as near as can be ascertained, from one to two years. Using three pairs of rolls in the above assumed 20 ton plant, which will actually run 20 hours per day during 250 days, crushing yearly 100,000 tons of crude ore, and estimating the average life of the tyres at one and a half years, one set of new tyres would, at this rate, be required for every 50,000 tons of ore crushed. The cost of tyres in this case will therefore amount to 0·4 cents per ton.

The following tests were made by Mr. Krom for granulation and dust. A sample of Port Henry ore ground in a Sturtevant mill to pass a 10 mesh screen ($\frac{1}{20}$ inch diameter hole) gave the following results:—

	Per Cent.
From 10 to 100 mesh (granular)...	86¼
Through 100 mesh (dust)...	13¾

and when ground to pass a 16 mesh screen ($\frac{1}{32}$ inch diameter hole):—

	Per Cent.
From 16 to 100 mesh (granular)...	82½
Through 100 mesh (dust)...	17½

Mr. Krom's opinion confirms Mr. Sahlin's remark that the percentage of dust in the sample of the ore operated on by him was too low, owing to the removal of the dust previously by the exhaust fan at the works.

Crushing the same ore with rolls to pass a 10 mesh screen Mr. Krom found:—

	Per Cent.
From 10 to 100 mesh (granular)	96¼
Through 100 mesh (dust)...	3¾

and when crushed to pass a 16 mesh screen this sample gave:—

	Per Cent.
From 16 to 100 mesh (granular)	93¾
Through 100 mesh...	6¼

These results collectively corroborate the well-known universal experience that grinding dry by attrition produces more dust than crushing with a system of rolls, and that most machines working on the former principle, owing to the high speed necessarily maintained, show greater wear and require more power than rolls do.

The Buchanan granulator is intended to occupy an intermediate place between a large breaker and rolls or other reduction machinery which

completes the crushing of the ore.* The receiving capacity of this granulator is $3\frac{1}{2}$ by 30 inches, or 5 by 30 inches, weighing respectively 10,000 to 12,500 lbs., and have a capacity of 4 to 8 tons per hour reduced to $\frac{1}{4}$ or $\frac{3}{8}$ inch ring-gauge. The machine is intended to reduce the coarse fragments from the jaw-crusher which do not pass the grizzly or screen below the breaker.

The jaws being very wide in proportion to their opening are made double, each being 15 inches in width, and work alternately to reduce the strain on the frame, which is heavy and solidly built. The distance between the jaws which regulates the size of the product can be adjusted by two steel tension-rods which pass through heavy lugs at the top of the frame and through holes cast in a movable tail-block and take the strain. The moving jaw which is pivoted below has an adjustable stroke, as the back toggle bearing can be moved up and down by two hand wheels in a recess cast on the face of the tail-block. The hand wheels pass through yokes or brackets below the tail-block, of which the latter form a part and can be secured by jamb nuts above and below the brackets. If the movable blocks are elevated the angle of the toggles will be increased, but if lowered it will be lessened and the throw of the swing jaw is shortened, preventing packing, whilst making a very fine product. The distance between the jaws and the length of stroke can both be regulated while the machine is in motion. The machine is provided with grease cups fitted with spring covers, a rubber spring and rod to assist the jaw on its back stroke, and the toggle bearings and toggles are chilled at the seats and points to ensure longer wear.

First-motion and Geared Rolls.

With regard to first-motion rolls, Prof. Egleston† says :—"In considering their efficiency and capacity (alluding, of course, to dry-crushing) we must take into consideration that the whole of the surface of the faces of the rolls is fitted to act on the ore, since they are even and parallel and wear so. The ore escapes from the rolls by gravity, just as soon as it is crushed fine enough to fall through the space between them. If the feeding is automatic, or even when it is not, there can be no clogging of them possible."

Theoretically, no doubt, these contentions are all true; but practically the writer does not think the wear can be said to be invariably even and parallel in the two tyres, especially if used for coarse-crushing, and proof

* *Engineering and Mining Journal*, New York, vol. lv., page 323.

† *Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., page 216.

that they do clog (or what amounts to the same thing) will be given later on. The professor goes on to say:—"A 30 stamp mill, with stamps weighing 850 lbs., dropping 90 times a minute, with shoes and dies 8 inches in diameter, which have in round numbers 50 square inches of surface, will have (50 by 90 by 30 =) 135,000 square inches of surface acting on the ore every minute. It may be considered that two sets of 22 inches rolls will have an average diameter of 21 inches. If these rolls make 80 revolutions per minute and are 14 inches face, they will have a contact surface of 141,120 square inches per minute, or a little more than a 30 stamp mill. If the diameter is increased to 30 inches, taking the average as 28 inches, and the length of face as 16 inches, and the number of revolutions at 80, the surface capacity will be equal to 47 stamps, and at 100 revolutions to 60 stamps. These calculations are made on the supposition that the surfaces acting are of equal efficiency in both stamps and rolls, and for that reason more favourable to the stamps than actual practice shows. With rolls properly constructed the pressure is constant at each instant of time,* while with stamps, on account of the varying height of the ore in the mortar, the cushioning of the stamp against the ore, and the fact that the stamp must not only crush but also force the ore through the screens,† it never can be constant. That the work can be done as well with rolls, which are very compact and take up only a little space, as with stamps, which occupy a much larger one, is shown by the experience of the mills where they are used. That they can crush as fine, is shown by the fact that rolls are now being used to crush after the stamps."‡

The condition of the ore particles leaving the rolls is such, being mostly angular, that to reduce the whole of the stone to the same absolute degree of fineness, it would be crushed with stamps through a finer mesh than the screen of the rolls taken for comparison.

When a movable pillow-block is used to support one of the rolls to prevent any pounding action, and keep them the right distance apart, they should be tied together with tension-rods and spiral-springs, the sliding roll being hinged so that it can swing back freely if anything passes between the tyres and overcomes the strain of the spiral-springs,

* This is only true, the writer thinks, so long as the space between the tyres is absolutely and exactly constant, which is not the case when they are subjected to uneven wear.

† In this connexion the action of the water in washing the ore out should not be overlooked. See also Prof. Munroe's fourth contention, page 328.

‡ It is not less true, however, that stamps are almost invariably used to crush after rolls in Germany, where theory, if anywhere, is carried to its finest point.

but their construction should be solid enough to prevent much vibration. The springs should not bear on the main tension bolts, as in such cases considerable force would be required to adjust the rolls. It is best to enclose the springs, therefore, in a nest between two washers, and to draw these washers together with special bolts, so that the whole will act as a rigid inelastic block until the maximum straining pressure is reached, when, for the first time, full compression will take place and relieve the rolls.

A set of 26 by 15 inches rolls, driven at 100 revolutions per minute, will reduce 20 to 25 tons of magnetic ore from $\frac{5}{8}$ inch to 10 mesh per hour. To hold rolls running at this rate it is absolutely necessary to use springs, as the old-fashioned lever and counterweights would be constantly on the jump. The springs before leaving the shop are compressed with their washers to a strain of 15 tons per square inch. The self-contained box makes it possible for a man with a small wrench to adjust them to any degree of fineness. The tyres should be seated on a conical core, on which they are drawn tight by jamb-bolts. To promote evenness of wear, the narrower the rolls are within reasonable limits the better, and a high-speeded narrow roll of large diameter in this respect is superior to a slow-speeded long roll of small diameter.

The most important part of a set of rolls are their tyres. Many makers use chilled iron or high carbon steel, but the writer's experience is in favour of using rather soft forged or rolled open-hearth steel.

The capacity of rolls is theoretically proportional to the superficial area of crushing surface passing the line of contact in a unit of time, and Mr. Sahlin is of opinion that a peripheral speed of 600 to 700 feet per minute will be found practicable. The writer, however, would suggest that if the diameter of the rolls is not proportioned to the speed the ore is broken partly by concussion, and not carried round and crushed by steady pressure, as it ought to be.

It is important that the axle of movable and rigid rolls should never become oblique under any application of strain, and it is a good arrangement to attach the movable journal boxes to a pair of strong levers firmly keyed into a heavy shaft, so that any excessive strain on one will cause both bearings to yield uniformly. The bearings of rolls should be large, well protected from dust, and perfectly lubricated.

For driving rolls at high speeds belts and pulleys are preferable to the use of geared wheels; the latter wear rapidly, and are rigid when in motion. To avoid slipping, both rolls should be driven by belts, but a greater amount of power may be advantageously applied to the rigid roll.

Adjustable scrapers set on the apron of the feeder should be employed to regulate the feed of the ore at any given point of the face of the rolls, as this is the best means of keeping the tyres true. A shaking trough or overfall flanged pulley are probably the best types of machine to employ as feeders.

Elevators show wear more rapidly than any other part of a crushing plant. Bucket elevators are to be avoided as much as possible; link-belt iron chains and screw conveyors have both proved decided failures for handling large quantities of ore. Leather or rubber belts to which steel or malleable iron buckets are riveted are more durable, but when practicable the best course seems to be to use horizontal or inclined carrying belts, travelling over concave carrying rollers, for elevating and conveying; a system largely in use in the corn warehouses of Liverpool, where the writer has seen it in operation. The greatest inclination at which a belt will effectively elevate is, however, only 27 to 30 degs., and as a consequence great length of belt is requisite to obtain high elevations.

Screening surfaces should err on the side of ample, in fact excessive, capacity. They are also exposed to a considerable amount of wear, and it is therefore necessary to construct them so that the screen plates can be readily changed without removing the frames. In dry crushing a hexagonal prismatic shape is, in the opinion of Mr. Sahlin, the best. The screen shaft should be placed in adjustable bearings, so that the time required to screen a given amount of ore can be regulated to the demands of the rest of the plant. Mr. Sahlin is in favour of using perforated steel plates with oblong diagonal slots fastened by wire-staples to the inside of rectangular hard wood frames, which can be readily clamped on to the cast iron frame. A better plan perhaps is to slide them into grooves in the ends of the arms. The rectangular shape of the frames admits of their being reversed before the nearest end of the screen, which suffers most wear, is cut through, increased service being thus obtained from a set of plates, and the oblong shape of the slots allows foliated crystals of minerals like hornblende and mica, on which the rolls make little impression, to pass the screen without clogging it. Woven wire screens have a tendency to clog, and their durability is less than steel-plate.

The hoppers below the screens should be lined with sheet-iron, or light bars of flat iron set at intervals (the spaces between the bars soon get filled with fine ore, which protects the bars as well as the wood casing).

In guaging the power required to run machinery very variable results may, as before pointed out, be found under different circumstances, and therefore investigations of the sort should extend over considerable periods of time.

With machinery running on a certain class of ore, its efficiency will apparently vary at times as much as 50 per cent., simply through slight changes of velocity, speed of feeding and discharge, or size of the material fed and delivered to the crushers.

Mr. Stetefeldt, in a paper on improved processes for the lixiviation of silver-ores, draws a comparison between Krom rolls and a stamp-battery. He states that the application of rolls is most favourable when the character of the ore permits comparatively coarse-crushing, without interfering with good roasting.

Assuming the crushing capacity of two sets of 26 inches Krom rolls to be equal to a 30 stamp-battery, with stamps of 850 lbs., dropping 7 to 8 inches, 94 times per minute. With two sets of rolls 100 tons of ore can be crushed in 24 hours,* all passed through a 16 inches wire-mesh, consuming not over 4 cords of wood. The fuel required for running 30 stamps is about 6 cords in 24 hours. Assume the following prices :—Freight, 1½d. per lb.; lumber, £10 8s. 4d. per 1,000 feet; wood, £1 5s per cord; carpenters, 18s. 9d. per day; and millwrights, £1 5s. per day. Certain items of construction—conveyors, elevators, screens, and dust-chambers—are about the same as in a dry-crushing mill. The building, however, will be smaller than for a battery, saving not less than £312 10s. in construction. Finally, the rolls requiring less power, a saving of at least £260 8s. 4d. will be made in purchasing and erecting engines and boilers in a mill with rolls.

A 30 stamp-battery, the plant including hardwood screen-frames and guides, wooden pulleys on cam-shafts, Tulloch feeders with iron-hoppers, and all necessary bolts, weighs 90,600 lbs., and costs in Chicago £1,218 15s. According to a statement of Messrs. Fraser and Chalmers, the framing takes about 36,000 feet of lumber, and the expense of setting up is reckoned at £833 6s. 8d. Hence the total cost of erecting a 30 stamp-battery, as specified, is :—

	£	s.	d.
Plant and foundry	1,218	15	0
Freight	566	5	0
Lumber	375	0	0
Erection	833	6	8
	£2,993	6	8
Extra cost of building	312	10	0
Extra cost of engines and boilers	260	8	4
	£3,566	5	0

* Here again a great deal depends on the character of the rock and nature of the ore.

The weight of one set of 26 inches rolls is 12,000 lbs., and costs in New York £468 15s. There is only one self-feeder required, and its weight is estimated at 2,000 lbs., and cost £41 13s. 4d. From these figures we deduce the cost of erecting two sets of 26 inches Krom rolls:—

	£	s.	d.
Plant at foundry	979	3	4
Freight	162	10	0
Cost of erection, including lumber...	145	16	8
	£1,287	10	0

The wear and tear of two sets of tyres, calculated on 20,000 tons of stone, crushed at the Geddes and Bertrand mill, taking the full capacity of the rolls at 100 tons, passed through a No. 16 screen, is estimated as follows:—

	£	s.	d.
Two sets of steel tyres at New York	159	3	4
Freight on 3,264 lbs., at 1½d. ...	20	8	4
	£179	11	8

Wear and tear per day of 24 hours:—

	£	s.	d.
Steel tyres	0	14	4½
Other parts (screens, supplies, and lubricants)	0	7	3½
Wages for repairs	0	5	2½
	£1	6	10½

Mr. Stetefeldt arrives at the following figures of cost for wear and tear of stamps, in a 30 head battery per day of 24 hours:—

	£	s.	d.
In all parts subject to wear and tear, supplies, screens, and lubricants*...	2	7	11
Wages	1	2	11
Total cost of wear and tear of stamps	3	10	10
Wear and tear of rolls...	1	6	10½
Showing a difference in favour of rolls of	£2	3	11½

In comparing expenses of running rolls and stamps, the interest and redemption of the excess capital, required in the construction of the original plant for stamps, cannot be neglected. Considering the short life of most silver-mines, this item should not be taken at a lower rate than 15 per cent. per annum. If we take the running time of the mill at 350 days in a year,† and consider a mill with stamps will cost £2,278 15s. more than one with rolls, the interest and redemption amount to 19s. 6d. per diem.

* Subdivided as follows:—Shoes and dies, 40 per cent.: tappets, boxes, cams, stems, cam-shafts, fingers, and boxes, 38 per cent.; screens, lubricants, screen-frames, guides, and carpenters' and machinists' supplies, 22 per cent.

† In Nevada they work on Sundays as well as on weekdays, or used to do so, though the practice is certainly inexpedient and undesirable.

Summarizing the above, we find the following daily saving by using 26 inches rolls, as compared with stamps :—

	£	s.	d.
Wear and tear and repairs	2	3	11½
Interest and redemption of capital ...	0	19	6
Fuel, 2 cords at £1 5s.	2	10	0
Total	£5	13	5½

Admitting the correctness of some of these figures, when applied to different cases, for example, supposing a stamp-mill to be run by water-power (as is often the case), the last item would obviously have to be excluded, and thus the advantage to the credit of rolls would at once be reduced to £3 3s. 5½d.

Then, again, as regards the extra cost of the buildings, the writer scarcely thinks the comparison altogether fair towards stamps. If you take merely the stamps and rolls, without the pans, settlers, furnaces, leaching-vats, and other apparatus necessary for saving the silver, it may hold good, but his own opinion is (till the contrary has been proved by actual figures), that if these are included, the tendency towards cost of building is precisely the other way.

The supposed saving can only be in width of building, but if it is a dry-silver amalgamating-plant, the room occupied by the pans will involve the same space cross-ways in the pan-room, whether rolls or stamps are used, and if, as Prof. Egleston remarks,* "the rolls are usually arranged in sets of two [as they ought to be and frequently are]. One of these receives the coarse ore from the crusher, which, after passing them, falls on a screen of determined size. What does not pass goes to a second pair;" it will add very considerably to the height of the mill, an addition which can be proved to be proportionately far more costly, than a saving in width.

Comparing the plans of some lixiviation-mills with those of dry-crushing pan-mills, it almost appears as if the latter class of building, as regards framing, was the more elaborate of the two. Take for examples the arrangement of a lixiviation-plant, illustrated in Mr. Stetefeldt's work on lixiviation, and a dry-concentrating roller-mill, in Galena Cañon, Nevada. This latter (speaking from personal acquaintance with it), the author ventures to say, is an extremely elaborate structure indeed, far more so (owing to its great height) than any pan-mill he has ever seen. A section of this dry-crushing plant is given by Dr. Raymond in one of the United States Government Reports.

* *Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., page 215.

Prof. Egleston adds to his previous statement quoted above this noteworthy remark: "When very fine crushing is required, three pairs of rolls can be used; but generally two sets are sufficient." This means that, in some cases, you can double, in others almost treble, the cost of plant at the foundry, and count on freight for four to six sets of rolls instead of two, whilst making a considerable addition as well to Mr. Stetefeldt's estimate for the wear and tear on the two or four additional sets of tyres, and check-plates, as the case may be. Under such circumstances, the saving of £2 3s. 11½d. and 19s. 6d. for wear and tear, and interest and redemption of capital, may be found to be really negative quantities.

Mr. Munroe gives the wear and tear of eight pairs of Cornish rolls, 14 by 30 inches, at Bonne Terre, after crushing 224,303 tons of magnesian limestone lead-ore in 1886-87 as under:—

	£	s.	d.
Seven pairs of chilled tyres, at £25	175	0	0
Gear-wheels and pinions	69	15	10
Total	£244	15	10

About £30 12s. 0d. per pair per annum. The tyres of the rolls, he remarks, used for coarse-crushing, are not turned down when worn, but are replaced by new ones, a statement which would indicate that there is more waste of metal in the use of rolls than stamps, and the author has frequently noticed the worn-out shells of rolls lying about the yards of works. It is owing, he presumes, to their considerable bulk and weight, that it does not pay (even if of iron) to break them up or dispose of them as they stand. The tyres of the fine-rolls at the Bonne Terre works are kept in good condition. Dr. Peters remarks,* "That the hardest chilled iron may be turned with an ordinary tool without difficulty, if a sufficiently slow motion is made use of in the process." In this way a set of chilled iron shells may be preserved in condition for fine crushing for a much longer period than usual.

For the screens connected with the rolls at Bonne Terre 21 sets of perforated plates, at £12 13s. 1½d. = £265 15s. 7½d., were required, or an average of 2·6 sets per year for each pair of rolls.

Each set of steel tyres on the rolls, therefore, lasts about a year, and a set of screen-plates about four months.

Prof. Egleston, again speaking of the experience at the Geddes and Bertrand mill,† states that the ore was crushed so as to pass a 15 to 20

* *Modern American Methods of Copper Smelting.*

† *The Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., page 487.

mesh screen; 30 mesh screens were first used, but it was found that the material did not discharge from these as well as from a coarser mesh, and that there was no necessity for treating the ore finer, as the roasting and leaching were better done on the coarser ore. Taken in connexion with what has been said before, the writer thinks this proves one of the points that he has emphasized—that rolls are not adapted to reduce ore in one operation to fine size. And this explains why they are not suitable for dry-pan amalgamation, because it means that either two (if not three) pairs of rolls must be used, one below the other, or extra work in the shape of grinding will be thrown on the pans. Added to this, there is the danger of a complete stoppage of a whole section of the works for a time, should a serious breakdown occur, and the time lost in changing tyres is not so very much less in the aggregate than changing stamp-shoes, as the latter is generally done when cleaning-up.

Messrs. McDermott and Duffield remark*: "Of late years rolls have been advocated for fine-crushing, and where the operation has to be performed dry, as a preparation for roasting with leaching, or chlorination, there is no question of the advantages to be gained.† For ores requiring wet treatment, however, free gold and silver and concentrating-ores, there is nothing but disadvantages connected with the use of rolls. Although there are various devices for concentrating and amalgamating dry-pulp, it is quite certain, from the failure of these processes in practice, though known for many years, that, where water is obtainable, dry treatment is not worth considering. The mere item of drying the ore, and the disadvantage of dust in dry-crushing—even if the process be granted the full efficiency of equivalent wet methods—are sufficient to prevent any serious consideration of dry-treatment, except as a matter of necessity from the absence of water. No one desires to dry an ore, crush dry, and then wet it again. For dry-crushing, stamps are not nearly so effective as for wet work, and, therefore, what would be a favourable comparison for rolls in one case becomes very different in the other. Rolls require screening and re-screening, with steady return of uncrushed particles, and all this means excess of water introduced, and the running of screens and elevators in addition to the crushing machinery. Again, the surface of roller-shells must be kept turned up true to do effective work in fine-crushing, or their capacity drops directly, while the capacity of stamps varies very little with wear of castings, as increased drop tends to neutralize

* *Gold Amalgamation and Concentration*, page 78.

† There is an exception to this mentioned by the writer, *Trans. Fed. Inst.*, vol. v., page 135.

decrease in weight. Some ores containing clay will press into thin cakes by the rolls, and become an increasing factor in the return from the screens."

The writer is not going however to deny, that under certain conditions, rolls are more suitable to employ than stamps, quite the reverse, as he has previously pointed out that rolls have most marked advantages over stamps, for certain special purposes. Nothing, in fact, too strong can be said in their favour, if it be intended to reduce an ore gradually in size, to an even grade, of coarse or fine sharp sand (1 millimetre or 0·039 inch and over), with a minimum production of slimes; and the whole method of Continental dressing, rests on this principle.

To obtain the mineral, in fact, with a minimum loss from comminution, the ore must be broken up, only just enough to unlock all the mineral it may contain, down to a certain size, and the mineral thus freed, must be separated from the mass at once; now whilst rolls naturally lend themselves to this class of treatment, stamps do exactly the reverse. On this account and others that have been pointed out, rolls find a wide application, in coarse-concentration, and lixiviation, in both of which evenness of product is essential for good work, and in sampling-works, attached to smelters. In fact, in most instances where one has to deal with the reduction of heavy pyritic ores, they justly hold the field, and in exceptional cases can be used in chlorination-works, though it is the more usual practice to treat free-gold ores in a stamp-battery.

Passing over the unusual case of the native copper amygdaloid, of Lake Superior, for which steam stamps are found best adapted, the author ventures to lay down four cases for which rolls are generally unsuited :—

1. Where you have to deal with a concentrating-ore like some grades of German lead-ore or Cornish capel, in which the mineral is finely disseminated (German, *eingesprengt*) or sprinkled through the gangue, it can be better liberated, either by stamps or roller-mills.
2. A clayey ore.
3. Any ore which requires, for reasons other than the one first stated, to be reduced in one operation, from a comparatively large lump-size, to fine powder.
4. Where amalgamation of any kind is to follow directly after comminution, especially when large quantities of ore carrying finely disseminated free-gold, have to be manipulated wet.

The writer will not comment on the first and third cases, as he has already referred to them. With regard to a clayey ore he alludes, of

course, to an ore containing a considerable proportion of clay, like some gold, zinc, and iron-ores, in which case the difficulty can only be overcome by passing the stone through a special cleansing-apparatus, generally a conical revolving-drum, 15 to 18 feet long, and 6 to 12 feet in diameter, fitted sometimes with inside blades, which break up the clay, allowing it to be washed off by a strong stream of water playing on the inside of the machine.

If the ore carries only a little clay, and the stone comes to the mill dry, it does not matter so much; if damp, it is advisable to run the rolls and screens wet, using a water-spray as they do at Clausthal. Whilst an excess of water does no harm but rather good, a little moisture in the stone itself, say 3 to 5 per cent., produces the effect of making it sticky, and is most pernicious.

In regard to the fourth case, the writer has already endeavoured to show the unsuitability of rolls for dry silver-amalgamation, and if this be granted, they are doubly so in most instances, for a wet free-gold process,* as it means the addition to the plant not only of elevators, screens, etc., but mechanical agitating and amalgamating-machinery as well, which, when the practical question of ultimate cost comes to be considered, reverses every other advantage to be gained. Added to this, no mechanical agitator, the writer believes, has yet been devised, which will give the natural splash of the stamps, which is as essential to plate-amalgamation as the turn of the wrist is to panning a gold prospect properly.

The writer may instance in proof of his contention what happened at the Haile mine, South Carolina, citing as his authority a paper by Mr. Theo. Blake.† The ore is a tough quartzite, carrying a large percentage of sulphides, which, according to Messrs. A. Thies and Wm. B. Phillips,‡ is associated with talcose slates. Whatever free-gold there is, is extremely fine, and is got out now by stamping in a battery, provided with inside and outside copper-plates. The pulp, which contains the balance of the precious metals present, is thence led on to Embrey end-shaking tables, and the concentrates thus obtained (representing about 9 per cent. of the ore) are worked, as previously described, by the Thies barrel-chlorination

* The question may be asked how this statement is reconciled with the Pestarena practice. The author may point out that the conditions of that particular case, like those of Mount Morgan, Queensland, are unusual, since the gold in the pyrites appears to be unusually free, the stone is heavily mineralized, and is subjected to a certain amount of concentration previous to crushing, a most unusual procedure, which is only practicable, owing to the low price of labour in Italy.

† *Trans. Am. Inst. Min. Eng.*, vol. xvi., page 753.

‡ *Ibid.*, vol. xix., page 601.

process, which appears to have at length solved this much-vexed question. The history of the undertaking is a peculiarly instructive one. Rock-breakers, rolls, and amalgamators, coarse and fine rock-breakers, and Chilian mills, the Disignole process, and dry-crushing, having each, the writer believes, had their turn, and been proved previously unprofitable.

Prior to 1888, it appears that the plan adopted was to pass the ore through two Blake Challenge stone-breakers (20 inches by 10 inches and 30 inches by 5 inches) and two multiple-jaw crushers, and thence through two sets of 30 inches by 16 inches Krom swinging-block rolls, from which it passed over Atwood amalgamators, and finally the pyrites was concentrated on Embrey tables.

Now, according to Mr. Blake,* who appears to have taken the matter up about this time, large as the capacity of the Krom rolls was, they could not pass the material they were designed to handle dry (from ¼ inch to 40 mesh) through quick enough, which would appear to be a mechanical indictment against the rolls: that it is not so, is due to the point already noticed, that two sets of rolls, one coarse and one fine, ought to have been used to give them quite fairplay. In fact if rolls require constant attention, and the ore does not pass through them properly crushed, you might for all practical purposes it seems, just as economically employ two sets at once, as incur the extra expense of wear and tear, coupled with the necessity of screening, re-screening, and elevating, the partially crushed products. If we follow Mr. Blake a step farther, he tells us, moreover, that they were running these rolls at 100 revolutions per minute, and this in itself was sufficient, the writer thinks, to account for their failure, to meet the call made upon them.

With coarse-crushing rolls general experience goes to prove that, in order to secure high tangential speed with a view to obtain large capacity, the diameter of the rolls must be duly proportioned to their rotatory velocity. This would be accentuated the greater the relative difference in size between the uncrushed and finished products, which in this case was extremely large. If speeded unduly high, the rolls will in fact refuse to bite ore-particles above a certain size, and you have as a result, a layer of ore between the rolls, dancing about like peas on a drum; in which case speed will produce just the opposite effect to that intended, and tend to choke them. The same also is true of a clayey ore, as a great deal of it will, in that case, be drawn through them in clots, causing extreme and uneven wear, and an enormous increase in returns of only partially crushed ore, so that an undue percentage has to pass through them again.

* "The Blake System of Fine Crushing and its Economic Results." *Trans. Am. Inst. Min. Eng.*, vol. xvi., page 753.

Mr. Kunhardt cites a case,* which actually shows that rotatory speed increases, it is true, the capacity of the rolls up to a certain point, but cannot be carried beyond given limits.

At one mill, where careful trials were made on the Continent, a pair of rolls 25½ inches in diameter and 11 inches face, required to run at 26 revolutions per minute in order to crush 15 tons of hard quartzose ore per hour, from 2 inches size to ⅔ inch. Reducing their speed diminished their output, but it was found, that without constant attendance, to shove the ore down between them (a remedy worse than the disease), the maximum speed they could economically be run at, was 16 revolutions per minute. The same difficulty, though to a less extent, may happen in fine crushing, and will be largely influenced by the difference in size between the ore fed to the rolls, and the size of the product they are required to deliver. The effect at Haile of running the rolls in the manner described was in fact precisely what one would expect. The belts constantly slipped, or were thrown, bringing the rolls suddenly to a full stop, and the surface of the shells or tyres had, in crushing 500 tons each, worn down ¼ inch, and become pitted and corrugated, while the wear on the screen-cloths actually amounted to 4s. 2d. per ton.

Mechanically, then, it was not surprising that the rolls jibbed, but that in no way alters the writer's proposition, viz.: that rolls are unsuited to wet gold-extraction, as a general rule; in fact, subsequent experience at Haile strengthens it, since they were superseded by a couple of Chilian mills, which Mr. Blake claims afforded a full solution of this difficulty, giving a capacity of 3¾ tons† per hour each (crushing the toughest quartzite of the Blauvelt mine), and as much as 10 tons of the softer decomposed ores, with a consumption of about 4 cords of wood per diem. The ore was crushed wet in this way. The two wheels of the mills each weighed about 1 ton, and were 4 feet in diameter, with 8 inches face, making 40 revolutions per minute. The wear of the tyres and segmental dies, estimated on 3,000 tons of ore crushed, was 0·12 of a pound per ton, and the cost of screens (screening the ore wet in a sort of raff-wheel) was reduced to (10 cents.) 5d. per ton. Yet, notwithstanding the question of capacity being solved, even this arrangement, was (it would appear from its abandonment) found too expensive as compared with stamps, for

* *School of Mines Quarterly*, series 2.

† At Pestarena, in Italy, Messrs. J. Taylor & Sons state that a Marsden 14 inches stone-breaker and Cornish rolls, 20 by 13 inches, are capable of passing through a sieve of 9 holes to the square inch, 1,000 tons of ore per month, working 10 hours per day, and 6 days per week; representing about this same quantity, 3¾ tons per hour, but crushing it much coarser.

direct amalgamation of the ore in bulk ; and as a result, the final change back to stamps had to be accepted as a commercial necessity.

In many low grade mines an increase in the tonnage crushed with an existing plant means not only a lower cost per ton of ore, but an increased revenue, which in some cases may make a profitless mine a success, and in certain cases, therefore, rolls may be profitably used for coarse-crushing before stamping it. The only mill constructed on this principle is the Huanchaca mill, at Antofagasta, Chili. By this means an increase of at least 20 per cent. in the capacity should be secured. It would not be out of the way either to have fine screens below the coarse trommels, so that the finer portion (no inconsiderable amount with many ores) might be sent direct to the pans or furnaces without passing through the battery, when the arrangement of the mill would admit of this, as was done with marked advantage at the Hermosa mill, Harshaw, Arizona.*

The ore should not, however, in such an arrangement be crushed smaller than $\frac{1}{4}$ inch by the rolls, and $\frac{3}{8}$ inch would be better, to avoid danger of choking the battery. Such an application of rolls might be utilized advantageously. Then again, dry-crushing by this means will always hold its own in lixiviation and some cyanide plants, because when the ore is crushed wet it must either be run into settling-tubs or direct to the leaching vats. In the former case extra labour is required, whilst in the latter the ore is deposited in layers according to specific gravity. The heavier and more rebellious minerals fall to the bottom, and the lighter particles forms a coating on the top which is almost impervious to the solution, and is consequently harmful. Finally, in connexion with dressing-machinery, rolls will never be superceded, but when the former is of a certain fixed capacity, and in direct connexion with them, it is a mistake for the crushing capacity of the rolls to be in excess. Should it be so, instead of gaining time and saving money by putting the ore through quickly, heavy loss is sustained by producing poor quality concentrates, or mineral is lost in the tailings by unduly forcing the capacity of the rest of the plant. A difficulty only to be got over by having storage bins, which in many instances could not be introduced without incurring unnecessary expense for re-handling or destroying the automatic sequence of operations which ought to obtain in all well-designed plants.

Bullion Refining, and Sampling.

When gold, containing iron (such as the crude bullion from mill-retorts), is being refined at the United States mints, Mr. R. P. Andrews†

* *Trans. Fed. Inst.*, vol. v., page 288.

†*Engineering and Mining Journal* (New York), vol. lv., page 54.

states that it is the practice to melt it in fire-clay crucibles placed in plumbago pots, with sulphur and carbonates of soda and potash, refining it afterwards with nitre. The surface of the melt is covered with a layer of iron sulphide slag, which may be removed by nitre, but much time will be gained if the melt is poured off into a mould, allowed to cool, and the slag broken off. This slag should be remelted with iron scrap for the recovery of any contained gold ; the bar gold is remelted and any gold obtained from the slag is added. The first refining is done by the addition of nitre, with occasional additions of borax-glass to thicken the flux. The flux is removed by means of a plumbago stirrer, which must be kept at the same temperature as the furnace. After the iron has been removed the lead and antimony are next got out by the oxidizing action of nitre. If these metals are in excess the successive additions of nitre will have little effect on them for some time. When the action has commenced its progress can be judged by the colours of the flux skimmed off, commencing with whitish yellow it passes to light green and then to dark green. Before the final removal of the lead rainbow colours pass rapidly over the metal as in cupellation. Towards the end of the refining the gold bubbles slightly, but as the last trace of lead disappears the surface becomes quiet and smooth as plate-glass. If the lead and antimony stubbornly resist the action of the nitre a little bichloride of mercury or sal ammoniac may be added. The chloride penetrates the whole body of the metal and forms volatile chlorides which rapidly rise and pass away. This action is different from that of nitre, as the latter remains on top of the metal and oxidizes the lead and antimony as they rise. Bichloride of mercury is more certain in its action than sal ammoniac, but it is more dangerous and therefore hardly to be recommended. Some refiners add oxide of copper to aid the extraction of lead, arsenic is removed by the same means as lead and antimony, but more time and close examination are necessary to get rid of it.

In the case of very base bullion, cupellation is almost a necessity. It can be effected in a wind-furnace connected by a wide throat to the flue of the other furnaces. The bottom of a plumbago melting-pot, well tamped with bone-ash, and hollowed out to receive the melt, will form a first-rate cupel. The cupellation requires no other attention than to keep the melt from freezing, and the process can proceed while other work is going on. If the metal should solidify before the impurities are all removed, their final removal can be effected by crucible refining.

Nitrate of potash is usually used in refining, though it contains 10 per cent. less nitric acid than nitrate of soda, the latter, however, is generally

impure and contains sodium chloride which, in the refining process, causes spurting and consequent loss of gold.

If the operator employs a plumbago pot for refining, the corrosive effects of nitre on the plumbago can be prevented by lining it with a bone-ash ring, which will gradually absorb caustic potash and become well fixed to the inside of the pot. This lining will protect the crucible and keep the nitre in the central part of the surface of the melt.

Silver is refined in the same way as gold, but extreme care must be taken to keep the temperature of the furnace low or silver will be volatilized. It is not advisable to use either bichloride of mercury or sal ammoniac in refining this metal.

The writer has endeavoured in this paper to steer clear of prejudice for or against any special make of machine or process, and wherever possible to express himself in general terms.

An engineer has obviously no more right to experiment in new machinery and methods of treatment on his own account with his employer's capital, without distinct permission to do so, than he has to accept a commission on the purchase of a machine as an inducement to recommend it. At the same time, be it said, the mine-owner not infrequently pursues the short-sighted policy of restricting the engineer in the matter of independent investigation, regardless of the loss to himself, and even goes so far as to offer direct inducement to the maker to produce inferior cheap work, by showing a total disregard of its quality, selecting plant sometimes on his own initiative, simply because it happens to be cheap. The author believes this interference is a most frequent cause of the failure of many mining undertakings, which is not to be wondered at, where the selection of plant and process is perhaps left to a person who has never seen a mine in his life, or at any rate knows nothing of the practical requirements of a business, so comprehensive in its nature, that a life devoted to it, and nothing else, is scarcely long enough to master more than its rudiments. It is undoubtedly to the interest of the mine-owner and maker alike, that the choice of mining-plant should be left to the engineer, who is responsible for erecting and running it ; at any rate, the person who assumes such a responsibility, should be personally acquainted with the mine and its local surroundings, in order to form a judgment of any value, upon such a matter.

It is the more important, because before you can alter a process that has once been established, however defective it may be, you must see your way to effect a saving which will compensate for the loss of capital sunk

in a first venture; in fact, the sum of the annual dividends everybody expects, and ought to get, when they put their money into a mine, largely depends on making a correct first choice, in selecting the means of treating the ore.

A battery improperly built, from a desire to do the work cheaply or carelessly, cannot be run at a high speed, but when well put together it may be run for a long time without damaging a bolt. The extra prime cost of a mill so constructed is a slight consideration if, as frequently happens, it enables a low grade ore to be treated at a profit which would otherwise be worked at a loss, as in the latter case a large capital sum is thrown away which could have been profitably employed by but a slight additional expenditure.

An engineer has to look at the business facts as he finds them; he is liable to make mistakes, probably, like other people, in doubtful cases, but with ordinary experience, judgment, and business prudence, he should be able to select the proper machinery for any particular purpose, such as will best forward the interests of those he serves—a fact mining companies might do well to consider more than they do.

Miners have a great deal to learn regarding the comparative merits of different structural materials used in machinery, applied to different purposes, the behaviour and comparative value, for instance, of mottled-iron (a mixture of 85 parts of hardest white-iron and 15 parts of tough grey-iron), ordinary chilled white-iron,* and ferro-alumina (a highly crystalline hard white-iron) for stamp-shoes and dies, as compared with ordinary cast-steel, chrome-steel, nickel-steel, aluminium-steel, and manganese-steel, or Hadfield steel, and for such a purpose evenness of wear, which involves uniformity of casting, is, the author thinks, rather to be aimed at than extreme hardness; or, to take another case, the comparative merits of open-hearth steel, which, though it costs about 5 per cent. more, is said to be a better material for roll-tyres than Bessemer-metal. Ordinary cast-steel for stamp-shoes lasts longer than iron, but costs proportionately more; on the other hand, a less frequent adjustment of drop is required with the more slowly-wearing material. In other cases it appears to be advantageous to use steel shoes and iron dies, as is done in some of the Californian mills, whilst in Australia excellent results are obtained with cast-iron shoes and wrought-iron dies. In districts remote from railways, and where freight charges are high, the first cost of material becomes, too, a matter of less moment than its durability.

* White-iron heads with a malleable-iron shank are much used. In dry-crushing wood wedges to fix the heads on with are apt to get loose, and the shoe drops off, they are therefore replaced by an iron strap $\frac{1}{4}$ inch thick.

If, for instance, it requires two sets of iron shoes and dies to do the work of one set of steel ones, it will evidently cost double as much in freight to keep the mill supplied with the former, and what would be the cheaper material at the foundry becomes the most expensive perhaps to maintain in use at the place where it is employed. This is the reason why chrome-steel is preferred at mines like the Mammoth in Pinal county, Arizona.

In many districts local foundries will recast worn-out iron shoes and dies, but are unable to produce steel-castings, and there is no market for old steel, while scrap-iron can generally be readily disposed of.

Speaking generally, it may be said that the wear of shoes and dies in a battery is affected considerably by the depth of discharge ; and the use of rock-breakers, grizzlies, and automatic feeders, by promoting even feeding, tend to produce evenness of wear. The best results are to be got not so much from the hardest material as by having the die made of metal more tough and less brittle than the die, examples of which have been given, or, to take a still simpler case, chilled and unchilled iron are also sometimes used. The wear of shoes, made of various materials, varies from 3·6 to 21·3 ounces of metal per ton of ore crushed, while that of the die has a minimum of 3 ounces and a maximum of 7·9 ounces. In the matter of cost the lowest is at the rate of 1d. and the greatest 3·8d. per ton of ore for the shoe, whilst for the die it runs from 2·8d. to 0·3d. The combined cost varies from 2·0d. to 6·5d. See Appendix A.

Again, the engineer requires to ascertain the relative quality of various kinds of timber, lubricants, etc., and in fact of supplies of all kinds, to compare their serviceability for different purposes, with their cost, information which is well worth collecting. Neither must he neglect improvements in tools, and small labour-saving appliances, which are constantly being introduced, tending to lighten the mechanical labour of the staff of the mine and works, and conduce to better or more expeditious, and consequently cheaper work.

It is a knowledge of the little wrinkles which save a few seconds time, not less than those which will save a penny on a purchase, or an extra fraction of a grain of the precious metals, which determine the economy or otherwise of management. New inventions and discoveries will no doubt be made in mechanics and metallurgy, and the dynamical applications of electricity will, at no distant date, have a most important bearing upon cheapening the cost of mining in districts where water-power is available as a prime motor, especially so in mountainous districts (like the Pyrenees for example) where local transport is of course difficult.

The author thinks, however, we must look for advancement in the future rather towards perfecting the details of the machinery and processes we at present possess, than to the discovery of new principles. We might substitute, for example, a small motor of the Diehl type for the fly-wheel belt and pulley of a rock-breaker or other machine, and drive it direct in this way perhaps with advantage in some situations.

Perfection is as unattainable in milling as in any other branch of industrial art, otherwise progress would cease. We can compare old methods with new not only with complete satisfaction at the advancement made, but with the consciousness that where so much improvement was possible, much room for improvement may remain.

The author regrets that he has recorded little that is new; he has drawn on scattered sources of information, and on the knowledge of others, viewed from the standpoint of personal experience, which is the only way he thinks to press forward towards a knowledge of the science constituting the important calling of the mining engineer, which, for diversity of interest, breadth of scope, and general utility (common features of all engineering), is surpassed by few, if any, of the paths in life that can be followed in this progressive age.

In conclusion, the writer wishes to acknowledge his indebtedness in particular to the *Engineering and Mining Journal* of New York, the writings of Mr. T. A. Rickard, and others for many valuable suggestions, which it is impossible in every case to individually acknowledge, though he has endeavoured to do so as far as lies in his power.

APPENDIX A.—Wear of Shoes and Dies in Stamp Mills.

Name of District.		Metal.	Weight. New.	Weight. Worn out.	Ore Crushed during Time of Service.	Metal Worn per Ton of Ore Crushed.	Cost of the Metal per Lb.	Value of the Scrap.	Cost per Ton of Ore Crushed.	Total Cost per Ton of Ore Crushed.
			Lbs.	Lbs.	Tons.	Ozs.	d.	d.	d.	d.
Gilpin Co., Colorado ..	Shoes	Cast iron ..	83	27	80	11·2	2	$\frac{1}{2}$	1·91	2·97*a*
	Dies	Do ..	48	26	78	4·5	2	$\frac{1}{2}$	1·06	
	Shoes	Chrome steel	111	31	202	6·3	4	..	2·20	3·58
	Dies	Do.	55	25	159	3·0	4	..	1·38	
Grass Valley, California	Shoes	Do.	152	48	251	6·6	$4\frac{1}{2}$	..	2·58	4·71*b*
	Dies	Cast iron ..	93	45	$96\frac{1}{2}$	7·9	$2\frac{1}{2}$	$\frac{1}{2}$	2·13	
Angel's Camp, California	Shoes	Chrome steel	175	40	585	3·6	$4\frac{1}{2}$	..	1·35	2·03*c*
	Dies	Cast iron ..	95	35	275	3·5	$2\frac{1}{2}$	$\frac{1}{2}$	0·68	
Mammoth, Arizona .	Shoes	Chrome steel	132	40	190	7·7	$5\frac{1}{2}$	..	3·82	6·57*d*
	Dies	Do.	120	37	240	5·6	$5\frac{1}{2}$	..	2·75	
Bendigo, Victoria ..	Shoes	Cast iron ..	180	38	115	19·7	$1\frac{1}{4}$	$\frac{1}{8}$	1·83	2·18*e*
	Dies	Wrought iron	98	26	335	3·4	$1\frac{1}{4}$	$\frac{1}{8}$	0·35	
Clunes, Do. ..	Shoes	Cast iron ..	196	56	105	21·3	$1\frac{1}{4}$	$\frac{1}{8}$	2·33	2·77*f*
	Dies	Wrought iron	138	30	420	4·1	$1\frac{1}{4}$	$\frac{1}{8}$	0·44	
Harrietville, Do. ..	Shoes	Fagot iron ..	172	38	185	11·5	$1\frac{1}{2}$	..	1·63	2·36*g*
	Dies	Do. ..	84	37	200	3·7	$1\frac{1}{2}$	..	0·73	
The Thames, New Zealand	Shoes	Cast iron ..	170	51	$135\frac{1}{2}$	14·1	$1\frac{1}{2}$	$\frac{1}{2}$	1·70	2·82*h*
	Dies	Do. ..	108	42	141	7·5	$1\frac{1}{2}$	$\frac{1}{2}$	1·12	

a No rock breakers or automatic feeders, ore moderately soft, lor *g* drop, wear of dies very variable.
b Rock breakers and feeders, ore very hard, dies contain $\frac{1}{8}$ steel scrap.
c Rock breakers and feeders, ore soft, short drop.
d Ore medium, rock breakers and feeders, no grizzlies.
e No rock breakers or feeders, ore variable, but medium hard.
f No rock breakers, feeders used, ore almost entirely quartzose.
g No rock breaker, feeders used, dies wear very irregularly.
h No rock breakers or feeders, ore of very variable hardness.

APPENDIX B.—Costs of Mills.

Black Hills.

	£	s.	d.
60 stamp gold-mill, containing 3 (15 by 9 inches) Blake crushers, 6 grizzlies 5 by 12 inches, 12 Tulloch feeders, 60 stamps of 850 lbs. each, 12 mortars, 12 electro-silver-plated copper-plates, 2 (30 inches) clean-up pans, 2 (10 inches) retorts, 2 amalgam safes, overhead crane, track, etc. All necessary shafting, pulleys, and belting, 18 by 42 inches Corliss engine, 2 (54 by 16 inches) tubular boilers, 30 inches tubular heater, No. 5 Knowles pump, etc. Approximate cost, erected	15,416	13	4
120 stamp gold-mill, containing 6 crushers, 24 feeders, all copper-plates, 2 (48 inches) clean-up pans, 1 (24 inches) retort, 120 stamps (900 lbs. each), 24 mortars, all shaftings, pulleys, and belting, 6 amalgam safes, 24 by 48 inches Corliss engine, and 4 (54 by 16 inches) boilers. Approximate cost, erected ...	51,041	13	4

Mexico.

40 stamp silver-mill, containing 2 (10 by 7 inches) crushers, 4 grizzlies, 40 stamps (750 lbs.), 8 feeders, 16 (5 feet) pans, 8 (8 feet) settlers, 2 (10 inches) retorts, 8 amalgam safes, 2 clean-up pans, 27 tighteners, all shafting, pulleys, belts, bolts, etc., and engines and boilers, etc. Approximate cost, erected	29,166	13	4
100 stamp gold-mill, containing 4 (15 by 9 inches) Blake crushers, 8 grizzlies, 20 feeders, all copper-plates, 2 (30 inches) clean-up pans, 20 mortars, 100 stamps (850 lbs. each), 22 by 48 inches Corliss engine, 4 (50 by 16 inches) tubular boilers, 1 (30 inches) heater, 1 ($3\frac{1}{2}$ by 16 inches) pump, 2 cranes, all shaftings, pulleys, belting, etc. Approximate cost, erected	41,666	13	4

APPENDIX C.—Comparative Table of Stamp Mills.

The following table shows at a glance the general characteristics of six of the principal gold-mining centres described by Mr. T. A. Rickard :—**

District.	Average Contents of the Ore.	Average Milling Costs.	Average Number of Stamps.	Weight of Stamps.	No. of Drops per Minute.	Height of Drop.	Depth of Discharge.	Crushing Capacity per Stamp.	Capacity of Mill.
	Dwts.	s. d.		Lbs.		Inches.	Inches.	Tons.	Tons.
Clunes, Victoria	9	2 5½	60	890	80	8	4½	2½	150
Ballarat	8½	2 4	40	950	70	8	3	2	80
Bendigo	9¼	2 5	40	900	72	9	3½	2½	90
Otago, New Zealand	10	2 11	20	800	77	7½	3½	1½	30
The Thames, New Zealand ..	9½	3 11½	30	700	70	8½	2½	1¾	52
Charters Towers, Queensland ..	25	12 6	30	950	74	8	2	2½	75

District.	Variety of Screen.	No. of Holes per Square Inch.	Percentage of Concentrates.	Average Gold Contents.	Retort Percentage.	Fineness of Bullion.	Life of Screens.	Loss of Mercury per Ton of Ore.	Consumption of Water per Stamp per Minute.
			Per Cent.	Ozs.	Per Cent.	Per 1,000.	Days.	Dwts.	Gals.
Clunes, Victoria	Perforated copper-plate ..	100	¾	3	40	970	25	⅛	8
Ballarat ..	Punched Russian iron ..	160	2	2	48	963	12	5	5½
Bendigo ..	Punched Russian iron ..	143	1	2½	50	955	11	7½	6½
Otago, New Zealand	Punched Russian iron and steel cloth ..	140	*	*	41	930	7	7	4
The Thames, New Zealand	Punched Russian iron ..	160	†	†	42	665	6	‡	§
Charters Towers, Queensland ..	Perforated charcoal iron ..	225	‖	‖	35	790	5	¶	¶

* Pyrites not saved as a rule. † Very variable; saved by blankets and treated in pans. ‡ Mercury usually attained a loss of 12 to 20 dwts. § Very variable. ‖ Very variable; from 1 to 70 per cent. and 2 to 20 ozs. value. ¶ Differ widely, according to number of pans and concentrators.

APPENDIX D.—Jaw-crushers.

Besides the crushers described in Part I. of this paper, some engineers favour a parallel motion by means of links, but as it is necessary in such cases to greatly increase the weight of the moving parts the vibrations will either become excessive or the speed must be reduced with consequent loss of output. It has been customary in the larger crushers to make the jaws in sections moving alternately. This reduces the vibrations but complicates the construction and increases the wearing parts, both vital defects with machines of the reciprocating jaw type. The writer favours the plan of pivoting the oscillating jaw, below the face plates, on the Dodge and Krom principle. By this arrangement the product is finished very evenly, the travel of the jaws is shortest, and the leverage greatest at the point where most work requires to be done. The working speed of a 10 by 20 inches Krom crusher is 400 revolutions per minute. The swinging jaw is made hollow to decrease its weight, and the face plates are made of rolled steel bars placed edgeways.

** "Gold-milling in Australasia."—*Engineering and Mining Journal* [New York], vol. lvii, page 101.

APPENDIX E.—Errata.

Vol. xlii., page 102, line 9, *for* " (carrying less than 6 ounces," *read* " (carrying not less than 6 ounces."

Vol. xlii., page 106, footnote, *for* " The idea Mr. Pearce intends to convey," *read* " The idea, it appears, Mr. Pearce intends to convey."

Vol. xlii., page 111, line 2, *for* " ratio to one another," *read* " ratio to the item for labour."

Vol. xlii., page 122, footnote, *for* " with 6 to 8 per cent. solution," *read* " with 0·6 to 0·8 per cent. solution;" and *for* " containing 2 to 4 per cent. of cyanide," *read* " containing 0·2 to 0·4 per cent of cyanide."

Vol. xlii., page 126, line 32, *for* " more particularly as regards cost, which precludes," *read* " (more particularly as regards cost) in situations which preclude."

With reference to remarks made on page 158, vol. xlii., Mr. Hamilton Smith wrote the author on September 25th, 1893, as follows:—" Brückner's furnaces were first used at the Alaska Treadwell mine, and changed to Spence furnaces, but with results not quite so good, costs being larger with no better extraction. In 1890 the Spence furnaces were abandoned, and long, single hearth, reverberatory furnaces were built, which have since been in use. Their capacity is 4 tons per diem each, allowing for repairs. With the Spence furnace, costs per ton were fully $18, and the gold extracted from the concentrates was under 80 per cent.; with present hand-washing the costs are from $8 to $9 per ton, and extraction about 90 per cent.; 95 per cent. could be got, but with large amounts of lime in the concentrates a 90 per cent. extraction is perhaps more economical than 95 per cent. would be. It was practically impossible with the Spence furnace to get uniform dead-roasts. After making futile attempts to remedy the defects of mechanical roasting the new management abandoned the system. Where 3 or 4 per cent., or perhaps 2 per cent., of sulphur can be left in, I imagine they will answer."

Writing again on September 30th, 1893, Mr. Hamilton Smith remarks:—" I think the chances are that no mechanical furnace will ever prove so successful at a mine for perfectly roasting sulphides as a reverberatory. The sulphides vary from week to week, so that hand-stirring in an open fire will always be more efficient. At large works where concentrates from various mines are mixed under the charge of a competent man, it is possible mechanical stirring may answer, but at one mine the owners cannot afford to pay very high salaries, consequently the man in charge of the furnaces and chlorination is generally (almost invariably) a man who works by rule of thumb. The most bothersome variation in the Alaska concentrates is the quantity of lime they contain. The more lime the larger the amount of salt required, and with considerable salt the volatilization of gold becomes notable, since we think the economical results are fully as good by increasing the salt with the contents of lime. In the old days chemists did not believe in a notable loss of gold by adding comparatively small amounts of salt in roasting, but experience clearly shows the old idea is wrong. The Frue vanners have probably been more successful at Alaska Treadwell than anywhere else, the tailings generally assaying less than 50 cents per ton leaving the vanners."

Vol. xliii., page 39, Mr. T. A. Rickard, in a letter to the author, dated December 28th, 1893, states that the plant of the North Star mill, Nevada County, has been enlarged since the statistics from which the information given in Table I. with regard to it were compiled. It now includes a 6 feet wheel, 4 feet and 3 feet Pelton wheel, with no engine, 3 rock-breakers, 40 stamps, 12 Triumphs and 4 Frue concentrators.

Vol. xliii., page 44, Table II. The South Clunes mill, Victoria, results are for 1890, the total milling cost is 2s. 3d. and not 2s. 11d., the difference being due to the cost for material, fuel, and repairs being 8½d. and not 1s. 4½d. as stated. The Saxon mill. Thames District, results are for 1892, they correspond with the statement from which they were taken, but ought to be transposed, being 1s. 2½d. for labour and 2s. 10½d. for supplies, the total of 4s. 1d. remaining unchanged.

Vol. xliii., page 45, Table II. (continued). Hidden Treasure mill, Colorado, insert opposite water *nil*. Mr. Rickard gives the following working costs of the North Star mine in California for the years 1888, 1889, 1890 :—52,708 tons were crushed, labour cost 1s. 3d., material and supplies 9d., water 1s. 3d., total 3s. 3d., or without water 2s. per ton.

Vol. xliii., page 47, line 8, *for* " Table I.," *read* " Table II."

Vol. xliii., page 48, third line from bottom, *for* " Table I.," *read* " Table II."

Vol. xliii., page 49, the labour data to extend Table III. so as to include the North Star mill would be :—1 amalgamator, 12 hours at 16s. ; 1 amalgamator, 12 hours at 12s. 6d. ; 1 rock-breakerman, 10 hours at 10s. 6d. ; 1 mechanic, 10 hours at 12s. 6d. ; and 1 vanner attendant, 10 hours at 10s. 6d.

Vol. xliii., page 50, line 26, *read* " £120 ; on the most unfavourable supposition we might fairly attribute," etc.

Vol. xliii., page 52, the results of practice referred to in Gilpin Co. are taken from Mr. Rickard's article on " Variations in Milling."

Vol. xliii., page 56, footnote, Messrs. McDermott and Duffield's *Gold Amalgamation*, page 5.

Vol. xliii., page 68, line 2, *for* " Ingo," *read* " Inyo."

Vol. xliii., page 70, line 24, *for* " Blaney," *read* " Blamey."

Vol. xliii., pages 70 and 71, footnotes * and †, *for* " see pages 68 and 69," *read* " see page 70."

Vol. xliii., page 133, line 6, *for* " 10 37 ounces," *read* " 10·34 ounces."

Vol. xliii., page 133, line 7, should read " above assay value of 1·74 ounces of gold and 10·22 ounces of silver, representing," etc.

Vol. xliii., page 133, line 13, should read " per ton amounting on 4·032 tons to 14s. 1d," etc.

Votes of thanks were accorded to the authors of the papers and the meeting terminated.

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
APRIL 14TH, 1894.

MR. A. L. STEAVENSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the previous General Meeting, and reported the proceedings of the Council at their meetings on March 31st and that day.

The proceedings of the Council of The Federated Institution of Mining Engineers were also reported.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. WILLIAM REMSEN APPLEBY, Professor of Mining and Metallurgy, University of Minnesota, 911, Fifth Street, S.E., Minneapolis, Minnesota.

Mr. RICHARD G. BELL, Engineer, 14, Archbold Terrace, Newcastle-upon-Tyne.

Mr. NICHOLAS JOHN BITZOS, Mining Engineer, c/o Mr. A. G. Sourlas, Balouk Bazaar, Constantinople.

Mr. ALBERT FREDERICK CALVERT, Mining Engineer, The Mount, Oseney Crescent, Camden Road, London, N.W.

Mr. HUGH BROWN COLLINS, Mining Engineer, Garscaddin, by New Kilpatrick, Dumbartonshire.

Mr. TAKUMA DAN, Mining Engineer, Miike Coal Mining Co., Chikugo, Japan.

Mr. ARTHUR DICKINSON, Mining Engineer, Warham Road, South Croydon, Surrey.

Mr. JOSEPH DOBBS, Colliery Owner, Jarrow Colliery, Castlecomer, County Kilkenny.

Mr. URIAH DUDLEY, Mining Engineer, Broken Hill Proprietary Co., Broken Hill, New South Wales.

Mr. HENRY CAWOOD EMBLETON, Mining Engineer, Methley, near Leeds.

Mr. A. GOERZ, Mining Engineer, P.O. Box 1961, Johannesburg, Transvaal.

Mr. FRANCIS HODGE, Mining Engineer, 7, Roskear Villas, Camborne, Cornwall.

Mr. EDWARD HOOPER, Mining and Civil Engineer, c/o Mr. J. H. Hooper, College Precincts, Worcester.

Mr. Henry Ryan Lewis, Mining Engineer, 7, Drapers' Gardens, London, E.C.

Mr. Thomas Charlton Renwick, Colliery Manager, Lumley Colliery, Fence Houses.

Mr. Claude Vautin, Metallurgical Chemist, 42, Old Broad Street, London, E.C.

Mr. Alexander Houstonne Ward, Mining Engineer, Coppull, Chorley.

Associate Members—

Mr. Robert Spencer Hawkins, Mining Engineer, 11, Selhurst Park, South Norwood, London, S.E.

Mr. James Hicks, M.S.A., Architect and Surveyor, Redruth, Cornwall.

Associates—

Mr. Harry F. Coux, Assistant Surveyor, Fairfield House, Spennymoor.

Mr. William Crown, Overman, 39, Fifth Row, Ashington Colliery, Northumberland.

Mr. Robert Alfred Harle, Back Overman, Hebburn Colliery, Hebburn, R.S.O., County Durham.

Mr. Frederick Charles Swallow, Assistant Manager, Wardley Hall, Newcastle-upon-Tyne.

Students—

Mr. Edward Aston Allport, Mining Student, Colliery Office, Medomsley, R.S.O., County Durham.

Mr. Edward Waters, Mining Student, Throckley Colliery, Newcastle-upon-Tyne.

The following gentlemen were nominated for election :—

Members—

Mr. Phillip Argall, Mining Engineer and Metallurgist, P.O. Box 1095, Denver, Colorado, U.S.A.

Mr. William Lawrence Austin, Ph.D., Metallurgist, P.O. Box 941, Denver, Colorado, U.S.A.

Mr. L. N. Britten, General Manager, Wolhuter Gold Mining Co., Limited, P.O. Box 494, Johannesburg, Transvaal.

Mr. Charles Augustus Harrison, Civil Engineer, North-Eastern Railway Co., Newcastle-upon-Tyne.

Mr. Robert Smith Heap, Draughtsman and Manager at Engineering Works, 2, Nelson Square, Burnley.

Mr. Michael Heslop, Manager, Rough Lea Colliery, Willington, County Durham.

Mr. James Hosking, Mining Engineer, Roskear, Camborne, Cornwall.

Mr. R. Kondo, Engineer, 7, Setomoncho, Nihonbashi, Tokio, Japan.

Mr. John Henry Johns, Mining Engineer, General Manager of the Ferreira Gold Mining Co., Limited, P.O. Box 1021, Johannesburg, Transvaal.

Mr. Charles E. Lawford, Mining and Mechanical Engineer, P.O. Box 238, Johannesburg, Transvaal.

Mr. William Matthews, Engineer, c/o Messrs. Black & Co., No. 5, China Bazaar, Calcutta, India.

Mr. William Moses, Mining Engineer, De Beers Consolidated Mines, Limited, Kimberley, South Africa.

Mr. Alfred Thomas Metcalf, Mining Engineer, General Manager of the Forbes Reef Gold Mining Co., Darkton (*via* Ermelo and Lake Chrissie, Transvaal), Swaziland.

Mr. René Proust, Mining Engineer, The Shamrock Lead Gold Mining Co., Limited, Samson's Flat, Westport, New Zealand.

Mr. Boverton Redwood, Consulting Adviser in reference to the Petroleum Industry, Glenwathen, Church End, Finchley, London, N.

Mr. Arthur Henry Reid, Civil Engineer, P.O. Box 746, Johannesburg, Transvaal.

Mr. George Robinson, Colliery Manager, Harton Colliery, South Shields.

Mr. Daniel Rowe, Mining Engineer, Ferreira Gold Mining Co., Limited, Johannesburg, Transvaal.

Mr. John John Thomas, Engineer and Manager, Slate Quarries, Newnham Cottage, Kendal.

Mr. Harvey Veasy, General Manager, Barrakar Coal Co., Limited, Barrakar, Bengal, India.

Mr. Henry A. Vezin, Mining and Mechanical Engineer, P.O. Box 256, Denver, Colorado, U.S.A.

Mr. John Huntley Young, Colliery Manager, Wearmouth Colliery, Sunderland.

Students—

Mr. George Charles Hague Davey, Mining Student, Manvers Main Colliery, Wath, near Rotherham.

Mr. John Denton, Mining Student, Netherseal Colliery, Burton-upon-Trent.

DISCUSSION ON THE "REPORT OF THE PRUSSIAN FIRE-DAMP COMMISSION."*

Prof. P. P. Bedson said that the report as it appeared in the *Transactions* was practically a condensation of numerous reports and investigations which had been committed to the hands of sub-committees appointed by the Commission. The report was a very exhaustive one, but he might be allowed to draw attention to one or two points which occurred to him as being of special interest. In the first place, dealing with the statistics as to the accidents in mines, it was well to note that amongst the accidents caused by fire-damp were others than those resulting from explosions of fire-damp. There were many cases recorded of accidents due to suffocation by fire-damp, and this seemed to him to point to the necessity of a lamp for miners which could be used not only as a source of light but also as a detector of fire-damp. The use of electrical lamps was desirable, on account of their possessing a greater illuminating power than

* *Trans. Fed. Inst.*, vol iii., page 1,105, vol. iv., page 631, and vol. v., page 500.

the ordinary miner's lamp or candle, but such lamps were useless for the detection of fire-damp or carbonic acid gas. He might also direct attention to the large number of analyses of fire-damp, from which it would be seen that fire-damp was not the simple material that some were inclined to regard it. These analyses showed that the combustible constituents of fire-damp were a mixture of marsh gas and hydrogen, together with some of the heavier hydrocarbon gases of the same series, and possibly hydrogen; the presence of these hydrocarbons appearing to be due to some specific properties of the different coals. Dr. Schondorff had observed that one easily inflammable fire-damp contained an admixture of carbonyl sulphide which made it especially inflammable; and in the gases met with in the Glückhilf colliery the presence of carbon monoxide had been established. The members would find in this report also an attempt to show that the hydrocarbon gases and the carbonic acid gas in coal have a certain relationship with the nature and composition of the coal, and that the gaseous contents of the coal are related in some way to its geological age. The report contains a valuable discussion on the various proportions of the several gases forming fire-damp, required to form explosive mixtures with air, and records some very interesting experiments by which the temperature of ignition of marsh gas and air had been determined.

Mr. M. Walton Brown said he might shortly summarize some of the recommendations contained in this valuable report. All mines were classed as fiery, wherein during the preceding two years fire-damp had been seen on a safety-lamp (3 per cent.). The use of mechanical ventilators was recommended with sufficient motive power in reserve, so that the usual air-supply might at any moment and without delay be increased by 25 per cent. Blasting operations in fiery mines were forbidden in the case of slowly detonating explosives, and in the case of dynamite and other quickly detonating explosives were only allowed in those districts where no fire-damp could be detected by means of an ordinary safety-lamp within a distance of 30 feet from the shot to be fired. The use of safety-lamps was compulsory in all fiery mines, and it was also ordered that numbered safety-lamps should be delivered complete to the workmen after being cleaned and repaired.

Mr. T. E. Forster asked if the report was one upon which any legislative action had been taken?

Mr. M. Walton Brown said the report had been acted upon throughout the kingdom of Prussia, the special rules having generally been revised in accordance with the recommendations contained in the report.

DISCUSSION UPON SIR ARCHIBALD GEIKIE'S PAPER ON "THE WORK OF THE GEOLOGICAL SURVEY."*

Prof. J. H. Merivale said he was inclined to agree with the writer and his critics that the Geological Survey of the coal-fields should be pushed on more rapidly, but this necessitated the spending of public money, which was no doubt voted entirely on the ground that it was for practical purposes. So long as some of the coal-fields remained unsurveyed the Geological Survey would no doubt continue to exist, but so soon as the work was done he was afraid that the Survey might come to an end. There were several reasons which rendered this undesirable, quite apart from the fact that they would not get interesting theoretical geological work carried out. A good deal had been heard of late years about technical education, and it appeared to him (Prof. Merivale) that the workshop was one of the necessary adjuncts of technical education, and the Geological Survey was the workshop for geologists. When the Geological Survey was finished, he did not know where geologists would get a practical insight into their duties. When the Geological Survey of Great Britain was finished there would still be the Geological Survey of our colonies; and young men now got their technical experience in the Geological Survey and then went off to Africa and Australia and surveyed those countries. Then again, where were the teachers of geology to receive their training. It was a fact that, with probably only two exceptions, all the leading teachers of geology in this country had obtained their training, or the greater part of it, when engaged upon the Geological Survey. Profs. Bonney and Lapworth were the only two exceptions so far as he knew, and he did not know where they would find such another workshop for the training of geologists when the Geological Survey came to an end. Sir Archibald Geikie had drawn attention to the important fact that, although the maps of many portions of the country that had been surveyed were not published, copies could be obtained on payment of a small sum.

Prof. H. Stroud read the following paper on "Magnetic Declination and its Variations," which was illustrated by lantern-slides :—

* *Trans. Fed. Inst.*, vol. v., page 142.

MAGNETIC DECLINATION AND ITS VARIATIONS.

By Prof. H. STROUD, M.A., D.Sc.

This paper is intended to explain the nature of the variations that occur in the value of the magnetic declination. The time variations may be grouped under the headings:—

1. Secular Variation.

This is a slow change in the magnetic meridian of the place. In 1657 at London the declination was *zero*, *i.e.*, the magnetic needle at that date pointed at London true north and south. The recorded observations before that date give easterly values for the declination; since 1657 the needle has pointed to the west of true north. The declination reached its maximum westerly value in 1816, when it was 24 degs. 30 mins. W.; the declination is now becoming smaller year by year, and at present just exceeds 17 degs. W. at London.

No explanation of the secular variation can as yet be given. It is noteworthy that the secular variation at the Cape of Good Hope has, unlike that at London, been in the same direction since observations were commenced in 1605. Profs. Rücker and Thorpe* point out that the secular variation is often appreciably different at neighbouring stations, as, *e.g.*, at Greenwich and Kew:—

	Declination at				Annual Variation at	
	Greenwich.		Kew.		Greenwich.	Kew.
	Degs.	Mins.	Degs.	Mins.	Mins.	Mins.
1880 ...	18	32·6	18	59·0	5·5	8·5
1881 ...	18	27·1	18	50·5	4·8	5·7
1882 ...	18	22·3	18	44·8	7·3	4·8
1883 ...	18	15·0	18	40·0	7·4	8·0
1884 ...	18	7·6	18	32·0	5·9	7·3
1885 ...	18	1·7	18	24·7	—	—

They find that the secular variation is greater in the northern than in the southern parts of the United Kingdom, and that the annual secular change was larger between 1857-1886 than between 1837-1857.

* *Phil. Trans.*, 1890.

2. Annual Variation.

The position of the needle depends to a very slight extent on the season of the year. Moreover the daily range (see below) is greatest in summer and least in winter.

3. Diurnal Variation.

The needle reaches its maximum westerly position at 1 p.m., and its maximum easterly at 10 p.m. (In the southern hemisphere east and west must be interchanged.) This variation is about 10 minutes, but goes through periodic changes, the period being, it is believed, the same as the sun-spot period of eleven years. The time of the maximum diurnal variation coincides with the time of maximum sun-spots. The diurnal variation is one and a half times as great at the maximum as it is at the minimum sun-spot epochs.

Prof. Balfour Stewart* draws particular attention to the close nature of the correspondence between sun-spots and declination (diurnal variation) as well as the lagging behind of the terrestrial magnetic effect.

The chief hypotheses that have been proposed to explain the diurnal variation are :—(1) The direct magnetic action of the sun upon the earth. But the time lag of the changes in the range of the daily variation of magnetic declination behind the corresponding solar changes is inconsistent with the theory of direct magnetic action; it seems to indicate rather some indirect influence caused by the radiant energy of the sun. (2) The action of earth-currents flowing in the earth's crust. (3) Prof. Balfour Stewart's suggestion that the cause of the diurnal variation exists in the higher regions of the earth's atmosphere in the form of a system of electric currents. We must remember that rarefied air is an electrical conductor, which, in moving in the earth's magnetic field, will cut lines of magnetic force, and thus will give rise to electric currents, and these, reacting on the earth's magnetism, will produce variations. We also note that air-currents are produced by inequalities of temperature, and, consequently, will be greater the more heat we receive from the sun, *i.e.*, will be greater when there are more sun-spots. The existence of the aurora, which is undoubtedly an electric current, favours this hypothesis. Moreover, Dr. Schuster† has applied the method, first suggested by Prof. Gauss, to decide whether the cause of the diurnal variation was inside or outside the surface of the earth. His results prove that the periodic cause of the variation is outside the surface of the earth. One difficulty

* *Encycl. Brit.*, "Meteorology." † *Phil. Mag.*, 1886.

arises in connexion with this explanation, viz., to explain how it is possible for such currents in the higher regions of the atmosphere to be one and a half times more powerful at times of maximum than at times of minimum sun-spots. Prof. Balfour Stewart has, however, pointed out that an increase in the radiation of the sun would mean not only an increase in the general radiation, but a special and predominant increase in such actinic rays as are probably absorbed in the upper regions of the earth's atmosphere.

4. Irregular Variation.

Besides the more regular changes before mentioned, the magnetic declination is subject to sudden disturbances of greater or less amount. When the variation is excessive, the disturbance is known as a magnetic storm. In the case of atmospheric weather we have regular differences of temperature morning and evening, but occasionally sudden disturbances occur which for the time mask the periodic changes; so in the case of magnetic weather we have regular variations of magnetic declination morning and evening, but occasionally sudden disturbances occur which overpower the periodic changes.

There appears to be some relation between sun-spots and magnetic disturbance, although the relation is probably not so intimate as between sun-spots and the declination (daily range). An oft-quoted example is that which occurred on September 1st, 1859,—coincidently with a most remarkable outburst on the sun's surface a disturbance of the needle took place. Mr. Ellis* has, however, pointed out that many greater magnetic movements than that of 1859 have since occurred, but no corresponding solar manifestation. The year 1892 was remarkable both for most violent magnetic storms and for great solar activity. The February magnetic storm of that year was the most violent experienced for ten years, and February, 1892, was also distinguished, not only by the number of sun-spots observed, but by the almost unprecedented dimensions of the largest among them. The magnetic storm commenced on the morning of February 13th, reaching the most important phase between 11 p.m. and 2 a.m. (February 14th). Both at Kew and at Paris the maximum disturbance was beyond the limits of the recording paper, so that absolute values of the maximum cannot be given, but a change of declination of 1 deg. 40 mins. was clearly shown at Kew.† The maximum disturbance at Paris was about 1 deg. 25 mins. The superintendent of the United

* *Nature*, November 9th, 1893. † *Proc. Roy. Soc.*, vol. liii., page 322.

States Naval Observatory confirms the remarkably sudden commencement of the storm at about the same time, the maximum disturbance being 1½ degs.

The following table contains the hourly positions of the needle, as taken from the Kew report*, for the days February 13th, 14th, and 15th, 1892, together with the diurnal range of the Kew declination, as derived from the hourly means of declination determined from selected quiet days in 1892 (winter and summer). The following figures indicate the salient points :—Mean westerly declination for 1892 being 17 degs. 36·7 mins.; the declination on February 13th, 6 p.m., was 17 degs. 16·7 mins.; 8 p.m., 17 degs. 50·7 mins.; 10 p.m., 17 degs. 58·7 mins.; midnight, 18 degs. 7·5 mins. February 14th, 1 a.m., 18 degs. 31·7 mins.; 3 a.m., 17 degs. 27·7 mins.

Hour.	Declination in 1892.						Hourly means of Declination during 1892.			
	Feb. 13th.		Feb. 14th.		Feb. 15th.		Winter.		Summer.	
	Degs.	Mins.	Degs.	Mins.	Degs.	Mins.	Degs.	Mins.	Degs.	Mins.
1 a.m. ...	17	34·7	18	31·7	17	39·5	17	35·2	17	36·0
2 ,, ..	17	35·9	17	56·7	17	38·4	17	35·3	17	35·7
3 ,, ...	17	37·7	17	27·7	17	38·7	17	35·2	17	35·4
4 ,, ...	17	38·9	17	38·2	17	38·7	17	35·3	17	34·6
5 ,, ...	17	38·9	17	52·7	17	40·6	17	35·3	17	33·3
6 ,, ...	17	39·7	17	40·6	17	38·9	17	35·3	17	32·2
7 ,, ...	17	33·7	17	38·7	17	40·7	17	35·1	17	31·6
8 ,, ...	17	32·1	17	35·2	17	38·7	17	34·6	17	31·7
9 ,, ...	17	35·1	17	34·7	17	36·7	17	34·8	17	33·2
10 ,, ...	17	24·0	17	34·5	17	37·7	17	36·1	17	36·2
11 ,, ...	17	47·7	17	36·2	17	39·2	17	38·4	17	39·4
Noon ...	17	36·7	17	38·2	17	40·7	17	40·5	17	42·7
1 p.m. ...	17	21·7	17	41·7	17	42·7	17	41·5	17	43·9
2 ,, ...	17	28·7	17	42·4	17	40·7	17	41·1	17	43·3
3 ,, ...	17	37·8	17	41·2	17	38·7	17	40·1	17	41·6
4 ,, ...	17	33·7	17	41·2	17	37·8	17	38·7	17	39·4
5 ,, ...	17	17·8	17	40·7	17	37·3	17	37·8	17	37·8
6 ,, ...	17	16·7	17	40·2	17	36·6	17	37·3	17	36·9
7 ,, ...	17	45·5	17	39 9	17	42·7	17	36·7	17	36·7
8 ,, ...	17	50·7	17	39·0	17	39·7	17	36·2	17	36·5
9 ,, ...	17	42·1	17	39·5	17	39·7	17	35·5	17	36·5
10 ,, ...	17	58·7	17	37·7	17	39·7	17	35·3	17	36·7
11 ,, ...	17	37·7	17	36·9	17	35·7	17	35·4	17	36·4
Midnight ...	18	7·5	17	41·1	17	36·7	17	35·5	17	36·1

In the case of the recent sun-spot of February 19th to March 2nd, 1894, which was much smaller than the February, 1892, spot, a magnetic disturbance, also smaller than in February, 1892, was noticed at 3·15 p.m., on February 20th, at Greenwich, the disturbance lasting 27 hours. After an interval of 24 hours another and more intense storm commenced and reached a maximum at 3 p.m., February 23rd.

* *Proceedings Royal Society*, vol. liii., pages 324 and 339.

There is almost certainly some relation between sun-spots and magnetic storms, but the exact connexion between the phenomena still remains unknown.

Major Dawson* has given an interesting table, deduced from the magnetic observations taken at Fort Rae in 1882-3, showing the number and magnitude of the disturbances in that year.

Mr. William Ellis, of the Royal Observatory, Greenwich,† has drawn attention to "the simultaneity of magnetic variations at different places on occasions of magnetic disturbance," and to "the relation between magnetic and earth-current phenomena." He points out that the beginnings of sudden disturbances are closely simultaneous at widely separated observatories.

In addition to the time variations above considered, it should be noted that magnetic surveys have indicated local disturbances which, in some cases, are of importance.

Profs. Rücker & Thorpe have recently completed a magnetic survey‡ of the British Islands, and Mr. Moureaux one of France. All who are interested in the British survey should refer to the original paper of Profs. Rücker and Thorpe. They have determined the magnetic elements (declination, dip, and horizontal intensity) at many hundred positions in different parts of the British Islands, and among their important results we may mention that which refers to local disturbances. A line, dividing regions of positive from those of negative disturbance of declination or of horizontal intensity so as to indicate attraction towards it, is called by them a ridge line. In like manner they call a line which separates the regions of influence of two attractive centres a valley line. They conclude that the local disturbances noticed during the survey are due to the presence of crystalline rocks, and especially of basalt, either visible on the surface or concealed by superimposed masses of sedimentary strata. Local disturbances have been investigated at Canna (an island near Skye), the Malvern Hills, in the neighbourhood of Reading, around the Wash, etc., and the various ridge and valley lines have been mapped out by them.

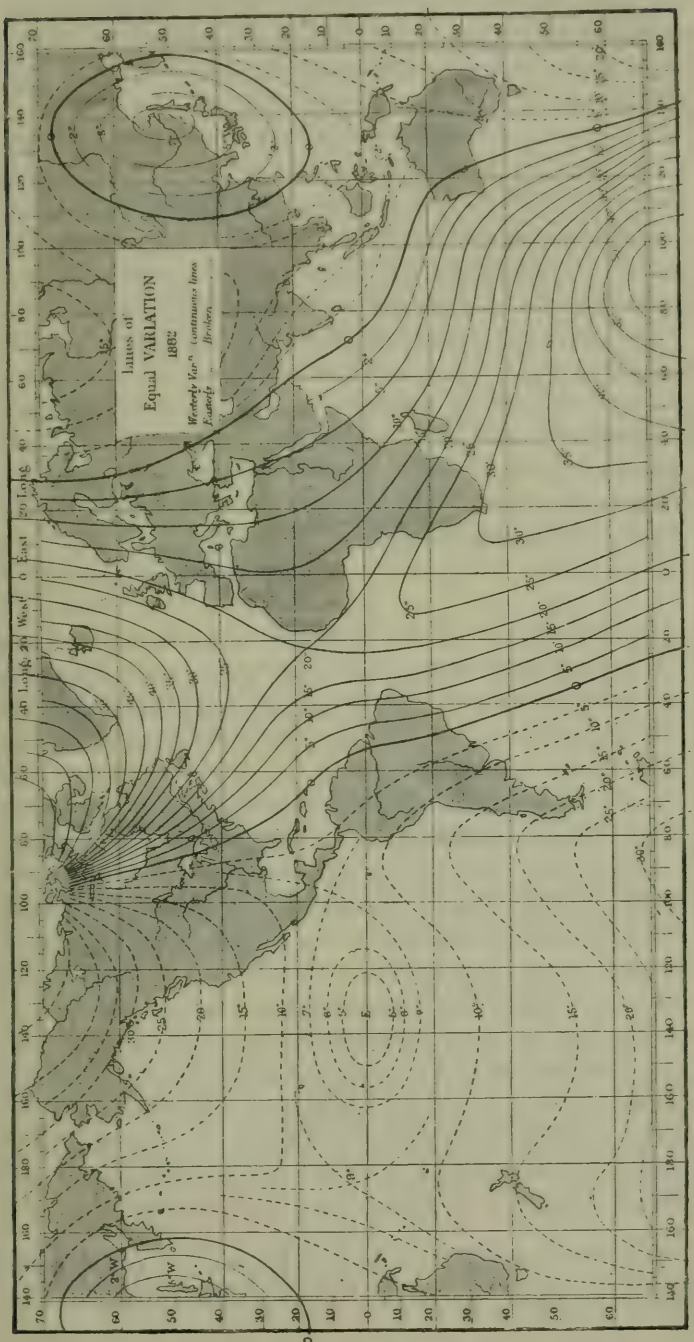
Commander Creak has shown that when islands disturb the magnetic needle in the northern hemisphere they attract, and in the southern hemisphere they repel the marked pole of the needle. This is what would be expected if they were the upper extremities of magnetic masses magnetized

* *Brit. Assoc. Report*, 1888, page 31.

† *Proc. Roy. Soc.*, vol. lii, page 191.

‡ *Philosophical Transactions*, 1890.

CHART OF THE WORLD SHOWING LINES OF EQUAL MAGNETIC VARIATION FOR 1882.



by the earth's induction. In Sweden the method of seeking for iron ore by means of the magnet has been carried to considerable perfection. It is based on the assumption that the upper parts of the beds of ironstone are of unmarked polarity by induction.

In some parts of the world very great local disturbances have been observed, thus in the neighbourhood of Bezout Island (off the north-western coast of Australia) a very great abnormal deviation was observed on H.M.S. "Penguin," in 1890. In one position the needle was deflected 55 degs. from its normal direction, and showed marked disturbances over an area of about a square mile. Again, in the south of Russia some remarkable anomalies have been observed: to give one striking example, the declination varied between 48 degs. west and 38 degs. east in the case of two places only 8 miles apart. The explanation of this anomaly may probably be found in the existence of some strongly magnetic peak at a small distance below the surface.

The annexed chart shows the lines of equal magnetic declination (termed variation in the chart) for the year 1882.*

The PRESIDENT said that Prof. Stroud's paper was full of interest and had clearly demonstrated the utter unreliability of the old circumferentor or miner's compass; it was an instrument entirely out of date, and should only be used by the overman for filling in the details of a trigonometrical survey.

Mr. H. JEPSON said that there was no doubt that the loose needle, when used to any considerable extent, was the cause of numerous errors. The difficulty arose in properly applying a suitable correction for the declination of the needle. By one method, the correction had been applied by putting the protractor on the meridian of the plan and slewing it round by an angle estimated to be sufficient for the declination of the needle for the lapse of time since the meridian had been drawn. Another method was to place the protractor on the meridian of the plan, and prick off the bearings with an addition for the correction of the declination. Sometimes meridians were drawn from time to time upon the plan, according to the variation of the declination. That was an unsatisfactory method, as in course of years the plan would become crowded with new meridians, some of which had probably not been very carefully drawn.

* Reproduced by permission of Captain E. W. Creak, R.N., F.R.S., by whom the original chart was drawn, from the *Elementary Manual for Deviations of the Compass.*

A more satisfactory plan was to enter on the margin of the note book each corrected bearing before commencing to plot the survey. In this way the correction is always applied, and it necessitated only a limited number of meridians on the plan. He recommended that records should be kept of the surveys made at collieries from time to time. When a new surveyor was appointed at a colliery he generally found considerable difficulties; no recorded surveys and not knowing what the last surveyor had used by way of correction for declination; and if he asked the colliery manager he was told in case of doubt that he had better re-survey from the shafts. Could Prof. Stroud state the degree of accuracy to be attached to the lines of declination shown on the map of England published annually by the *Colliery Guardian?*

Prof. STROUD wished to mention one point with regard to the subject of his paper. One was liable to think that the magnetic needle varied in a very erratic way; but it was necessary to remember the magnitudes of the variations considered. He thought that the declination of the needle, shown on the map referred to, might be relied upon in this district, at any rate within a few minutes, and any result obtained from its use would probably be correct within ten minutes.

Mr. G. B. FORSTER proposed a vote of thanks to Prof. Stroud for his interesting paper on the variation of the needle, a subject which must be of great importance to all mining engineers. When they examined many of the very old mining plans, they found that these were not so very far wrong; and if the needle were taken care of and carefully used it might still be of service, although they all knew the desirability of the main base-lines of a survey being obtained by some other system of surveying.

Mr. J. B. SIMPSON seconded the vote of thanks, which was cordially adopted.

The following paper by Mr. A. P. Wilson on "Minerals and Mining in Tasmania" was taken as read, and the meeting concluded.

MINERALS AND MINING IN TASMANIA.

By A. P. WILSON, Assoc. M. Inst. C.E.

Introduction.

Tasmania has not hitherto occupied a very prominent place as a metal-producing country; but among the many professional mining engineers and geologists who have recently been attracted to it by astonishing reports of mineral wealth, the almost unanimous opinion is that before many years have elapsed it will in this respect take a much more conspicuous position.

Roughly speaking, Tasmania may be described as an island of triangular shape, situated about 200 miles south of Melbourne, the capital of Victoria, having its apex towards the south and its base facing north, so as to form with the Victorian coast the passage known as Bass Strait. In area the island is about two-thirds the size of Ireland, and in its general topography presents a succession of hills and valleys, the former attaining in many instances a height of over 5,000 feet above sea-level.

A large proportion of the uncultivated land consists of virgin forests, densely timbered, and rendered impassable by thick undergrowth, those of the west coast being particularly difficult and impenetrable. The explorer is obliged to cut his way in such country by the free use of the axe, and path-cutting at the rate of $\frac{1}{4}$ mile per day is considered rapid progress.

It will readily be understood that prospecting under such circumstances is work of extreme difficulty, where vegetation so closely conceals the soil and impedes the progress of the investigator; and undoubtedly it is these physical obstacles that have so long retarded the exploration and development of the west coast, which was the first part of the island discovered by Tasman in 1642.

At the present time Tasmania claims at least one developed mine of world-wide fame—the enormously rich tin-mine of Mount Bischoff, and it is not improbable that in a few years' time the silver-lead field of Zeehan and Dundas will have attained similar prominence. The Beaconsfield gold-mine has obtained a reputation second only to Mount Morgan in the Australian colonies. Besides gold, tin, silver, and lead, other minerals, such as copper and bismuth, coal and iron, marble and slate, are known to exist, and are already worked to a limited extent.

Tin-mining.

The tin deposit at Mount Bischoff in the north-west of the island was discovered by Mr. James Smith in the year 1871. This discovery was the result of years of indefatigable hard work on the part of Mr. Smith, who was a most enthusiastic prospector. His wanderings alone for weeks and months in the impenetrable forests required the utmost courage and endurance, and his discovery of the Mount Bischoff deposit was a well-earned reward.

Nearly all the tin exported from other districts of Tasmania up to the present time has been derived from alluvial deposits; but in the case of Mount Bischoff the tinstone is obtained by opencast working on what is supposed to be a decomposed lode of enormous width. The tinstone occurs disseminated throughout the mass, which, as exposed in the Redface opencast (as it is locally called, from the prevailing colour) consists principally of brown and red oxides of iron, with occasional streaks of yellow ochre, the rest of the face being porphyry, both decomposed and as hard lumps. The porphyry is separated from the gossan by a thin belt of sand, and next to the porphyry comes slate of Silurian age.

The face from which the ore is at present taken measures about 200 feet in length by 80 feet in height, and is simply a quarry cut into the mountain-side, increasing in height as the working is carried farther into the hill. Beneath this bench the same ground has been proved to continue for a depth of 100 feet before encountering the bastard sandstone. True lodes, from 6 to 8 feet in width, as well as dykes of porphyry, all impregnated with tinstone, occur irregularly throughout this bastard sandstone.

The softer portions of the ore coming from the Redface are treated in sluice-boxes, whilst the harder portions are sent direct to a battery of 75 heads of stamps. After crushing, the ore is passed through classifiers and jigs to buddles and slime-tables, as in the usual process of ore-dressing. The Mount Bischoff mill is lighted by the electric light, so that it can be worked up to its full capacity of 300 tons of ore per day; and the average output is 250 tons of dressed black tin per month, which is sent to Launceston to be smelted. The mine is connected with the north coast at Emu bay by a railway 40 miles in length. The clean-dressed tin-ore, delivered at the smelting-furnaces, costs about £16 per ton.

The principal other tin districts of Tasmania are Ringarooma, Branxholme, George's bay, and Blue Tier, all in the north-eastern part of the island. The whole of the tin obtained in these districts is, as already

stated, derived from alluvial deposits found in river-terraces, frequently covered by Tertiary basalts, high above the level of the present water-courses. Mount Bischoff, however, produces more tin than all the other districts combined.

It is not unlikely that in the future, tin-mining will take a still more prominent position than now in the industries of Tasmania, for even the districts closely engaged in this class of alluvial mining have been very imperfectly explored, and it is more than probable that further examination will bring to light massive lodes of tinstone, which have been the sources of the alluvial ore now worked, the great strength of which may be inferred from the stones of solid cassiterite, weighing upwards of 100 lbs. each, which have been found from time to time.

Gold-mining.

The first undertaking to attract attention was the Tasmania gold-mine at Beaconsfield in the north of the island, near the mouth of the river Tamar. This mine, since the year 1877, has yielded over 300,000 ounces of gold, of the value of about £1,100,000. The reef has carried gold from the surface to the deepest level yet worked, viz., 450 feet, and a borehole which cut the reef at a depth of 700 feet produced a core showing gold freely, so that there is every prospect of the future of this mine being as brilliant as the past. The reef occurs in grits and sandstones of Lower Silurian age.

The principal quartz-mining districts are Lefroy, near the mouth of the river Tamar, and Mathinna, in the north-east of the island, which, during the past twenty-five years, have produced large quantities of gold, but have latterly, owing to financial depression, been rather neglected. In the latter district, however, an impetus has recently been given to the industry by the highly successful results obtained by the New Golden Gate Mining Co. This company is working in two more or less parallel reefs, which have an average width of about 4 feet, although in places the quartz measures over 20 feet in thickness. The gold occurs principally in the free state, but the quartz also carries about $1\frac{1}{3}$ per cent. of auriferous pyrites, which is saved by means of Frue-vanners, and the gold extracted from it.

The average yield of the quartz, apart from the pyrites, is something over 1 ounce per ton. The reefs occur in a Silurian slate; they are generally very much split up and broken near the surface, and do not pay to work until a depth of about 70 feet has been attained. The cost of working, including all expenses, is said to be £1 9s. $1\frac{1}{2}$d. or 7 dwts. 17 grains of gold per ton of quartz.

Of recent years several discoveries of gold have been made in the practically unexplored districts of the west coast, and nuggets, weighing as much as 143 and 243 ounces, besides many smaller ones, have been found. These explorations have been confined to the panning of gold in the streams and creeks by a few prospectors, and nothing has been done to systematically develop any of the finds.

Copper-mining.

One of the most important of recent discoveries is that of Iron Blow at Mount Lyell on the west coast, so called from the hill being capped by a mass of purple hæmatite, gossan, and barytes, which has proved to be the oxidized outcrop of an enormous deposit of iron and copper pyrites.

This formation is nearly 300 feet wide, and the oxidized portion is said to average 3 ounces of gold per ton. In 1892, the ore was treated for gold in a small stamp-battery, with amalgamated copper-plates, but, owing to the gold being very scaly, the larger proportion of it was lost.

Recently further explorations have proved the existence of enormous quantities of iron and copper pyrites beneath this mass of gossan, and the property has lately been reported on by Dr. Peters, of Boston, U.S.A., among others. He estimates that over 4,500,000 tons of this ore have been proved, and he puts down the average contents per ton as follows:—

Copper	100 pounds.
Silver	3 ounces.
Gold	2½ dwts.

In the course of exploration a rich zone of silver fahlerz was discovered and sunk on close to the footwall of the pyrites-lode. Up to September, 1893, 116 tons of this ore have been raised, containing 24¾ tons of copper and 134,774 ounces of silver. Specimens have been found containing over 16,000 ounces per ton, or, say, 50 per cent. of silver. This rich vein had evidently been derived from a leaching of the ore at a higher level, the silver and copper contents of which had again been deposited in a concentrated form lower down on the footwall. It is proposed to work this enormous deposit by opencast, and results equal to those of the famous Rio Tinto mine are confidently anticipated by those capable of forming an opinion.

Silver-lead-mining.

Tin and gold have thus been actively worked for a considerable number of years, but it is only recently that the enormous deposits of rich silver-lead on the west coast have become known.

Mount Zeehan is situated about half-way down the west coast of the island, about 9 miles distant from the sea at Trial bay. It was so named by Tasman after one of the ships of his expedition (the "Zeehaan" or "Sea-hen"), having been one of the first summits sighted by him in 1642.

The mountain is surrounded by undulating country, covered with extremely dense bush and impenetrable forest and scrub, and in view of the indescribable difficulties of contending with so rank a growth at a distance from any base of operations or supplies, it is little to be wondered at that the mineral wealth of the district has remained so long unknown and undeveloped. Hardy pioneers were not to be deterred, however, by the forbidding aspect of the country; they manfully struggled on against all difficulties and dangers in the pursuit of their laborious task, and were ultimately rewarded by the discovery of what promises to be a marvellously rich and extensive silver-field.

The formation in which the lodes occur is slate of Silurian age, and the presence in abundance of decomposed felspathic rock indicates, perhaps, the origin of the fissures which were subsequently filled with argentiferous galena in matrices principally of spathic iron, iron pyrites, and baryta.

The whole field, that is the ground pegged out into claims, comprises an area of upwards of 140 square miles, more or less covered by a network of galena-lodes, all of which was open to the first comers who might denounce their claims and pay the necessary fees.

In the case of Tasmania, neither the terms nor the necessary mode of procedure are onerous. The discoverer must peg out the ground claimed in square blocks of 20, 40, or 80 acres, each corner being marked with a wooden peg, at least 6 inches in diameter, projecting at least 3 feet out of the ground. A fifth peg has also to be placed as near the centre of the claim as possible, and upon this the claimant's name and the date of the pegging have to be marked. An application form (for which a fee of 20s. is charged) for the land is then filled up in duplicate, one copy of which is attached to the walls of the nearest post-office, and the other is sent to the nearest Government surveyor, who, after satisfying himself as to the situation and measurements, transmits the application to the Minister of Mines. (The survey fee varies from £3 12s., to £5 for areas not exceeding 20 acres, and from £8 15s. to £11 15s. for areas not exceeding 100 acres.) In the event of no protest being received or no other claimant coming forward, a lease is granted in due course on the condition that the claimholder shall pay 5s. per annum per

acre leased, and shall expend a certain moderate sum annually on the property. No royalty rent or other such charge is payable, and the lease holds good for twenty-one years, after which it is renewable on application. These provisions are applicable to all kinds of mineral leases, except gold claims, for which the terms are slightly altered.

It would be impossible within the limits of this paper to describe the whole of the Zeehan field, but the property which the writer went out specially to inspect may be taken as typical of the whole, including, as it does, some of the best galena-lodes yet discovered in the field.

The mineral occurs generally as bright, close-grained galena, for the most part cropping-out at the surface. According to report, it is due to this that the value of the field was first recognized. A prospector proceeding from the direction of the coast in search of tin ores having sat down to rest on the bank of a creek, chipped with the butt end of his axe a piece off a projecting rock at his side, and to his astonishment it revealed a bright, steely-like mineral, which a subsequent assay proved to be extremely rich argentiferous galena.

Some of the lodes have been traced for upwards of $\frac{1}{2}$ mile in length. The galena-outcrops occur for the most part in the lowest lying ground, and when the lodes course into higher ground the galena is replaced by a siliceous and, in some places, manganiferous gossan. This occurrence may signify that the ores in the low-lying ground, being under the permanent water-surface of the country, have been thereby protected from oxidation, and it may be that the gossan which at one time covered them has been removed by denudation, leaving the unaltered galena exposed.

For the most part, the galena in these lodes is free from any large proportion of deleterious minerals. Occasionally a little zinc-blende is found, or small quantities of sulphide of antimony, but silver is the principal metal found associated with the galena. In the gossan outcrops lead is found in the condition of argentiferous carbonate, while beautiful specimens of crocoisite or red chromate of lead are here also met with. In some of the smaller lodes specimens of horn silver (chloride of silver) and embolite (chloro-bromide of silver) have been discovered, but never as yet in any considerable quantity. The veins of decomposed felspathic rock which are also occasionally found, are peculiarly rich in silver, tons of what is known as kaolin ore carrying silver to the extent of 400 ounces per ton, having been already mined. When the galena is not solid the matrix in which it usually occurs is carbonate of iron and, to a less degree, iron pyrites, barytes, and quartz. The physical structure of the galena, though generally presenting a small close-grained fracture,

is frequently somewhat coarse-grained, but it never presents the large cubical flakes which are commonly met with elsewhere. The coarse-grained variety assumes at times a beautiful iridescence, not unlike peacock copper-ore, but having a prevailing tinge of blue, and such portions of the ore are more highly argentiferous than such other parts, even of the same stone, as do not display this feature.

The average result of many assays made in the field under the writer's supervision, taking ores both rich and poor, yielded 60 per cent. of lead and 65 ounces of silver per ton. By selecting the ore and laying aside for concentration by ore-dressing machinery such portions as carry too much gangue the assay value would be much increased. Already nearly 800 tons of galena, as it comes from the mine, have been sent to this country from one mine, and on an average it has assayed upwards of 66 per cent. of lead and 110 ounces of silver per ton, realizing here a net price of more than £20 per ton. From another mine on the field 2,224 tons of ore, carrying 221,349 ounces of silver and 623 tons of lead have been extracted and disposed of locally, realizing £36,038, or nearly £18 per ton.

Many of the lodes, however, carry too much gangue for direct exportation, and their ores require to be concentrated. Seven concentrating-mills have been erected on the field.

There are two series of lodes—one series coursing to the west of north and the other to the east of north. Sufficient development has not as yet taken place to enable a determination as to whether these two series were synchronous deposits, or were formed at different epochs, and which of the two is the older. Those lodes that have a north-westerly strike are, for the most part, larger and more permanent in length; but those coursing to the east of north are richer in silver and, so far as proved, quite as permanent in depth.

Parts of the field are low-lying and swampy, and as the rainfall of the Zeehan district averages about 100 inches annually, powerful pumping machinery is an absolute necessity for proving the lodes below the water level. This fact, together with the extremely insufficient means which have hitherto existed of communication with the coast have had the effect of greatly retarding the development of this very promising field.

Up to the early part of 1892, the only approach to the mines from the coast, except by bridle-paths, was a corduroy road—a cartway made of small trunks and boughs of trees laid side by side across the track to prevent the horses and the wheels of the carts sinking into the swampy ground. Carriage from the coast along this track to the mines, a dis-

tance of about 11 miles, used to cost £3 per ton for comparatively light articles, while for heavy pieces of machinery almost prohibitive sums were charged. The difficulties, uncertainties, and dangers of landing goods on a stormy and unprotected coast, added materially to the cost of freight.

The mines are now, however, connected with an excellent harbour, the port of Strahan, a distance of 29 miles, by a railway of 3½ feet gauge (the standard gauge for all Tasmanian lines), which the Government have built for the benefit of this mining field at a cost of about £200,000. This line was only undertaken after the Mount Zeehan field had been favourably reported on by experts employed by the Colonial Government.

Zeehan, at the time of the writer's visit, had some 200 to 300 inhabitants, most of whom were living in tents. Twelve months later the population had risen to over 4,000, and the tents of the early campaigners have been replaced by permanent buildings, including 8 hotels, 4 churches, 3 banks, assembly rooms, court-house, and all the necessary adjuncts of a colonial town. It is now the third largest town, as regards population, in the colony.

Unfortunately at the present time, Tasmania, in common with all the Australian colonies, is somewhat under a cloud of commercial depression, and the development of the Zeehan field has consequently been checked; many of the mines have been obliged to shut down, owing to the inability of the shareholders to pay the calls that are made for the opening up and development of their properties. However, this depression can only be temporary, and when fortune smiles once more on Tasmanian industries, the west coast will assuredly be heard of as a large lead and silver-producing district.

Coal-mining.

Coal is worked to some extent in Tasmania, but this industry is quite in its infancy. The coal that is being worked belongs, for the most part, to the Mesozoic period, and is generally high in ash, although containing a good proportion of carbon; it is used on the Government railways, and, when the firebars are suitably arranged for it, gives satisfactory results. The principal districts are Mount Nicholas on the east coast, and Sandfly and Jerusalem in the neighbourhood of Hobart, but outcrops of seams have been found in several other places. Nowhere has the coal been worked to any considerable depth beneath the surface, and whilst it is much faulted it is possible that, at a greater depth, it may improve in quality.

The following are averages of several analyses of Tasmanian coals :—

	Mount Nicholas.	Jerusalem.	Sandfly.
Water	6·49 ...	4·10 ...	4·51
Volatile matter ...	34·37 ...	20·60 ...	16·27
Fixed carbon ...	48·43 ...	57·40 .	66·66
Ash	11·60 ...	17·90 ...	12·18
	100·89	100·00	99·62

The annual production of coal is about 40,000 tons, none of which is exported.

Iron Ore-mining.

Iron ore occurs in abundance in Tasmania, principally as red and brown hæmatites. About twenty years ago a company was formed to work and smelt the large masses of hæmatite which exist in the vicinity of Ilfracombe, near the mouth of the river Tamar. Unfortunately, the attempt ended in failure, owing to the presence of chromium in the ore, which produced a metal too hard and brittle to be of any commercial value. It is remarkable that the discovery of this undesirable element in the ore was not made until after buildings had been constructed, a railway built, furnaces erected, and a sum of money amounting to £80,000 expended.

The Ilfracombe hæmatite occurs in beds of serpentine-rock, which carries veins of asbestos ; the ore contains from 55 to 70 per cent. of metallic iron, and the following are complete analyses made by Mr. Woodgate in the year 1880 for Mr. Johnston, the Registrar-General of Tasmania :—

Samples	I.	II.	III.	IV.
Volatile on ignition...	11·2 ...	19·5 ...	5·1 ...	12·0
Alumina	2·0 ...	15·3 ...	5·8 ...	2·4
Sulphur	trace ...	— ...	— ...	—
Phosphorus	trace ...	— ...	— ...	—
Zinc and magnesia ..	— ...	— ...	— ...	trace
Manganese	0·5 ...	— ...	— ...	0·4
Chromium	trace ...	3·0 ...	3·4 ...	—
Peroxide of iron ...	79·8 ...	42·5 ...	80·0 ...	70·8
Protoxide of iron ..	— ...	1·0 ...	— ...	—
Silica	6·1 ...	18·4 ...	4·8 ...	13·8
Unascertained ...	0·4 ...	0·3 ...	0·9 ...	0·6

The iron reduced from these ores contained as much as from 3 to over 5 per cent. of metallic chromium, and after some thousands of tons of metal had been produced and shipped to England, where it was experimented on by various chemists and manufacturers, the undertaking was abandoned. But although the efforts made to develop this promising mining industry failed at Ilfracombe owing to the presence of this undesirable metal, there is no evidence of the presence of chromium or other deleterious mineral in other extensive deposits, such as those occur-

ring at Mount Donaldson, Penguin, and other localities. It is therefore possible that in the future these may be developed and worked with satisfactory results, especially if, with improved means of transit, local coal becomes available.

Miscellaneous Minerals.

Bismuth occurs in large quantities in hornblende-rock at Mount Ramsay, between Mount Bischoff and Mount Zeehan. It has been described by Prof. Ulrich as "a discovery representing, to my knowledge, one of, if not the most important and richest made of this rare metal in recent times." The deposit has not as yet been developed in any way, on account of its inaccessibility, but efforts are now being made to open it up.

Nickel has been found in serpentine-rocks of the Heazlewood district on the west coast, but it is not yet known if it occurs in sufficient quantities to make it worth working. The ore is a hydrated double silicate of magnesia and nickel, similar to that worked in New Caledonia.

Sapphires are frequently found in the tin-drifts of the north-eastern districts, together with garnets, zircons, topazes, etc.

The limestone-cliffs on the Gordon river, of Silurian age, yield an excellent marble, which takes a fine polish. Very good hydraulic cement has been made from the limestone of Maria island, lying a few miles to the north of Tasman's peninsula, and its manufacture promises to become an important industry.

In the words of the Government statistician:—"A review of the mineral resources of Tasmania shows that she is bountifully endowed with great variety and abundance of mineral wealth. Her future prosperity depends more on the development of her stores of coal and metals than on the agricultural, pastoral, and timber industries. While these are not thought lightly of, it is remembered that in them our colony has no special advantages over her rivals, and in many respects is at a disadvantage, hence it is not to them that she will owe future eminence. The great wealth of minerals and natural advantages as a manufacturing centre will, however, necessarily make her take a leading part not only as a producer of raw materials, but also as a maker of finished products. With large coal-supplies, an excellent temperate climate, plentiful water, and large stores of gold, tin, iron, silver, and lead her manufacturing future is assured."

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD AT THE GRAND HOTEL, WHITEHAVEN, JUNE 21ST, 1894.

MR. A. L. STEAVENSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the previous General Meeting, and reported the proceedings of the Council at their meetings on May 26th and June 16th.

The proceedings of the Council of The Federated Institution of Mining Engineers were also reported.

The SECRETARY read the Balloting List for the election of officers for the year 1894-95.

ALTERATION OF BYE-LAW 17.

The SECRETARY read, on behalf of the Council, a proposed alteration of Bye-Law 17, which will be moved at the Annual General Meeting, on August 4th, 1894.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. PHILIP ARGALL, Mining Engineer and Metallurgist, P.O. Box 1095, Denver, Colorado, U.S.A.

Mr. WILLIAM LAWRENCE AUSTIN, Ph.D., Metallurgist, P.O. Box 941, Denver, Colorado, U.S.A.

Mr. L. N. BRITTEN, General Manager, Wolhuter Gold Mining Co., Limited, P.O. Box 494, Johannesburg, Transvaal.

Mr. CHARLES AUGUSTUS HARRISON, Civil Engineer, North-Eastern Railway Co., Newcastle-upon-Tyne.

Mr. ROBERT SMITH HEAP, Draughtsman and Manager at Engineering Works, 2, Nelson Square, Burnley.

Mr. Michael Heslop, Colliery Manager, Rough Lea Colliery, Willington, County Durham.

Mr. James Hosking, Mining Engineer, Roskear, Camborne, Cornwall.

Mr. R. Kondo, Engineer, 7, Setomoncho, Nihonbashi, Tokio, Japan.

Mr. John Henry Johns, Mining Engineer, General Manager of the Ferreira Gold Mining Co., Limited, P.O. Box 1021, Johannesburg, Transvaal.

Mr. Charles Edward Lawford, Mining and Mechanical Engineer, P.O. Box 238, Johannesburg, Transvaal.

Mr. William Matthews, Engineer, c/o Messrs. Black & Co., No. 5, China Bazaar, Calcutta, India.

Mr. William Moses, Mining Engineer, De Beers Consolidated Mines, Limited, Kimberley, South Africa.

Mr. Alfred Thomas Metcalf, Mining Engineer, General Manager of the Forbes Reef Gold Mining Co., Darkton (*via* Ermelo and Lake Chrissie, Transvaal), Swaziland.

Mr. René Proust, Mining Engineer, Mammoth Hydraulic Sluicing Co., Matakitaki River, Central Buller, New Zealand.

Mr. Boverton Redwood, Consulting Adviser in reference to the Petroleum Industry, Glenwathen, Church End, Finchley, London, N.

Mr. Arthur Henry Reid, Civil Engineer, P.O. Box 746, Johannesburg, Transvaal.

Mr. George Robinson, Colliery Manager, Harton Colliery, South Shields.

Mr. Daniel Rowe, Mining Engineer, Ferreira Gold Mining Co., Limited, Johannesburg, Transvaal.

Mr. John John Thomas, Engineer and Manager, Slate Quarries, Newnham Cottage, Kendal.

Mr. Harvey Veasy, General Manager, Barrakar Coal Co., Limited, Barrakar, Bengal, India.

Mr. Henry A. Vezin, Mining and Mechanical Engineer, P.O. Box 256, Denver, Colorado, U.S.A.

Mr. John Huntley Young, Colliery Manager, Wearmouth Colliery, Sunderland.

Students—

Mr. George Charles Hague Davey, Mining Student, Manvers Main Colliery, Wath, near Rotherham.

Mr. John Denton, Mining Student, Nethersea1 Colliery, Burton-upon-Trent.

The following gentlemen were nominated for election :—

Members—

Mr. R. Hay Anderson, Mining Engineer and Mine Manager, La Compania Unida Mejicana de Minas, Guanajuato, Mexico.

Mr. Harry Ashcroft, Assistant Manager, Borrea Coal Co., Limited, Seetarampore, Bengal, India.

Mr. William Henry Borlase, Mines Manager and Engineer, Brown How, Ulleswater, Glenridding, near Penrith.

Mr. Ian Cameron, Mining Engineer, Sudbury, Ontario, Canada.

Mr. James Wilcocks Carrall, Mining Engineer and Acting Commissioner of the Amoy Customs, Amoy, China.

Mr. Horatio Collins, Manager of the Woodbush Gold Mining Co., Ld., Haenertsburg, Transvaal.

Mr. S. T. Croasdell, Mechanical Engineer, Lowca Engineworks, Whitehaven.
Mr. George Matthew Darby, General Manager of the Katrass Kerria Coal Co., Ld., Katrass P.O., Barrakar, Bengal, India.
Mr. W. H. Furlonge, Mining Engineer, c/o Messrs. Proctor Bros., Antananarivo, Madagascar.
Mr. Strangeman Hancock, Mining Engineer, Rand Club, Johannesburg, Transvaal.
Mr. Guy Roche Johnson, Mining and Civil Engineer, Longdale, Alleghany County, Virginia, U.S.A.
Mr. Robert Kay, Colliery Manager, South Tanfield, Stanley, R.S.O., County Durham.
Mr. James Macnab, Engineer, 17, Bolsover Street, Portland Place, London, W.
Mr. J. M. Main, Mining Engineer, Whitehaven.
Mr. James Miller, Chemist and Metallurgist, The Mexican Gold and Silver Recovery Company, P.O. Box 121, Mexico City, Republic of Mexico.
Mr. Ralph Patterson William Oswald, H.M. Inspector of Mines, 10, Rose Bank, Whitehaven.
Mr. Joseph Provis, Lecturer on Mining, Metallurgy, and Technical Engineering, School of Mines, Kapunda, South Australia.
Mr. Beverley S. Randolph, Civil and Mining Engineer, Frostburg, Maryland, U.S.A.
Mr. William Redshaw, Colliery Manager, Whitehaven Colliery Office, Whitehaven.
Mr. William Schroller, Engineer and Contractor, 13, Old Elvet, Durham.
Mr. Elgin Scott, Manager, Ropienka Oil-Wells, Ropienka, Galicia, Austria.
Mr. Arthur Whitcomb Sheafer, Mining Engineer and Geologist, Pottsville, Pennsylvania, U.S.A.
Mr. Edward Thomas, Mine Surveyor, Mines Department, Sydney, New South Wales.
Mr. Edward Watson, Colliery Manager, Redheugh Colliery, Gateshead.
Mr. C. L. Waugh, Colliery Manager, Buckhill Collieries, Great Broughton, Cumberland.

Associate Members—

Mr. Arthur D. Carr, Land Agent, Whickham Rectory, Whickham, R.S.O.
Mr. A. E. W. Gwyn, Agent of the Ingersoll-Sergeant Drill Co., 114a, Queen Victoria Street, London, E.C.
Mr. H. R. Hancock, Moontana Mines, South Australia.
Mr. George C. Hooper, Surveyor of Mines, Whitehaven Collieries, Whitehaven.
Mr. Stephen Humble, Jun., Engineer, 9, Victoria Street, Westminster, London, S.W.
Mr. Reiji Kanda, Mining Engineer, c/o Mr. Takahira Kanda, 9, Awajicho-Nichome, Kanda, Tokio, Japan.
Mr. Ochiltree Macdonald, Editor of the *Canadian Colliery Guardian*, 161, Hollis Street, Halifax, Nova Scotia.
Mr. Frank L. Nason, 5, Union Street, New Brunswick, New Jersey.
Mr. Frederick Danvers Power, c/o Messrs. Henderson & Macfarlane, 2, Bridge Street, Sydney, New South Wales.
Mr. Umfreville Percy Swinburne, Salisbury, Mashonaland, South Africa.

STUDENTS—

Mr. CECIL ALFRED BURNE, Mining Student, Cornsay Colliery Office, near Durham.

Mr. JOHN RIDLEY HETHERINGTON, Mining Student, Cornsay Colliery Office, near Durham.

Mr. WILLIAM JONES, Mining Student, Trimdon Grange Colliery, Trimdon Grange, R.S.O.

Mr. HERBERT WILLIAM MIDDLETON, Mining Student, Trimdon Grange Colliery, Trimdon Grange, R.S.O.

Mr. JOHN WILLIAM PEEL, Mining Student, Coundon, Bishop Auckland.

AWARDS FOR PAPERS.

The SECRETARY read the following list of papers read during the year 1892-93, for which prizes had been awarded by the Council to the authors:—

"Use of Cement in Shaft-sinking." By Mr. Bennett H. Brough.

"Manometric Efficiency of Fans." By the Rev. G. M. Capell.

"Observations on Fans of Different Types Working on the Same Upcast Shaft." By the Rev. G. M. Capell.

"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment." By Mr. A. G. Charleton.

"Steam Boilers and Forced Blast: the Perret System for Burning Dust and Rejected Fuels; with Notes on Testing Boilers." By Mr. Bryan Donkin, Jun.

"The Gold-bearing Veins of the Organos District, Tolima, U.S. Colombia. By Mr. Edward Halse.

"Education of Mining Engineers." Py Prof. J. H. Merivale.

"The Geology and Coal-deposits of Natal." By Mr. R. A. S. Redmayne.

"Gold-mining in Brazil." By Mr. E. M. Touzeau.

Mr. J. D. KENDALL read the following paper on "Miss-fires":—

MISS-FIRES.

By J. D. KENDALL.

In working the hæmatite-mines of the Whitehaven district, where large quantities of powerful explosives are used, miss-fires are a frequent cause of injury to the miners, injuries that are sometimes fatal, and therefore the subject of this communication seems to the writer one which the Institute might very properly consider.

The principal explosives used in the Whitehaven district at the present time are nitro-glycerine compounds; of these gelatine-dynamite is by far the most largely used, but blasting-gelatine and gelignite are also employed, though in smaller quantities. The total quantity of these explosives used in the district annually, at the present rate of working, is about 110 tons, and the number of shots fired about 880,000. The number of miss-fires may probably be taken on an average at 6,700 per annum.

The usual method of charging the holes is to put the explosive into the hole first, the last cartridge having the cap in it, then to stem with a more or less clayey material to the top of the hole. In very wet ground water alone is usually employed for tamping.

Miss-fires arise either from defective materials or bad workmanship.

Defective Material.

Miss-fires from defective materials may arise as follows :—

(*a*) *Inferior explosives*, arising in many, if not most, cases from exudation of the nitro-glycerine. This is a defect, however, which is fortunately not of very frequent occurrence now.

(*b*) *Inferior detonators* are also met with occasionally, but not often.

(*c*) *Defective fuze*.—This is the most abundant source of the miss-fires which arise from the use of inferior materials. Even in fuze of first-class make there are sometimes found lengths in which for 2 or 3 inches not a trace of powder is to be seen, and sometimes the powder is of an inferior quality for short lengths. Fuze of either of these descriptions may be most dangerous. For even if it should eventually "run," it must "hang fire" more or less, which may so deceive the miner as to lead him back again into

his working merely to be shot. The touch-thread or threads burn at rates varying from $1\frac{1}{2}$ to 5 inches per minute in the open air, but confined in the fuze they will probably burn much more slowly, and be much more likely to die out. It is quite possible, too, that a deadening of the fire conveyed by the touch-threads may so occur that the rate of burning in a barren part of the fuze may be reduced to that at which the yarn enclosing the powder-train is consumed; that is, from 0·38 to 0·85 inch per minute in the open air. As confined, even this slow rate may be reduced to one-half; so that a piece of fuze barren for 3 inches may "hang fire" for a quarter of an hour. This to a miner, waiting for a shot to go off, is a very long time, and many cases have occurred in which he has gone back too soon and been seriously injured in consequence. The usual rate at which fuze runs is about 2 feet per minute, although in some cases it is much slower, only 1·4 foot per minute. Therefore from $1\frac{1}{2}$ to $2\frac{1}{2}$ minutes is sufficient for any ordinary length of fuze to run, and when this time is exceeded a miner concludes either that there has been a miss, or that the fuze is hanging fire. If he lack caution, or be very anxious about his work, he may go back too soon and suffer in consequence, as many have done. A miner should never return to a working in the case of a miss-fire, unless he has heard the cap explode, before the expiration of at least 20 minutes.

Fuze is no doubt, however, often blamed for miss-fires when the real cause is defective charging.

Defective Workmanship.

Miss-fires from defective workmanship may arise in several ways:—

(*d*) The detonators may be so insecurely fixed on to the fuze that in course of stemming the two may be separated, especially if the swab-stick be rather too thick and draw back the fuze during the operation of stemming. In such a case the fire may not reach the detonating-material. Before this could happen, however, the fuze must be pulled completely out of the detonator or it must be defective, so that it will not spit properly, to use the miner's expression; for when both the fuze and detonators are good the latter are exploded, even if the fuze only just enters the cap. It is a common opinion among miners that the fuze must be cut very square at the end, and then fixed in contact with the detonating-material, but that is not necessary. All that is needed is that the detonators shall be firmly fixed on to the end of the fuze, so that they cannot be separated in the course of stemming. It is of no consequence if the cap be pulled out of the dynamite; detonation of the latter will take place just the same,

provided, of course, that they are in contact. Some miners knowing this, often, in charging their holes, put in the explosive first and then the cap, with the fuze attached, on the top of it. It is a common but erroneous opinion that the cap must be placed within the explosive.

(*e*) In wet ground it is necessary to grease the cap, that is, to fill up the space between the sides of the cap and the fuze with tallow, so as to prevent water from gaining access to the inside of the cap. This is sometimes neglected or imperfectly done, and miss-fires are the consequence. The writer has had greased caps and fuze immersed in water for twenty-four hours, yet the fuze ran and the caps exploded, just as if they had not been in water at all. Caps with fuze attached, but not greased, would not explode after immersion for the same length of time. The explanation is probably that a comparatively thick film of water intervenes between the fuze and detonating-material in the ungreased cap, so as to prevent the fuze-fire from reaching the latter, because caps that have been immersed in water for three-quarters of an hour and afterwards drained explode in the usual way, and fuze so treated runs as usual, so that mere dampness is not sufficient to account for miss-fires of this description.

(*f*) The sawdust, with which the caps are filled as they leave the factory, may also cause miss-fires if it be not removed. The writer has made experiments in this direction. Caps that contain only sufficient sawdust to barely cover the detonating-material will explode, but $\frac{1}{16}$ inch of sawdust will prevent explosion. If the sawdust be not quite dry, it is most difficult, and in some cases impossible, to shake it out of the cap, and it is not advisable to attempt its removal by any sharp-pointed instrument, unless it be of wood: therefore caps in which the detonating-material is not visible should be rejected. The writer has tried to explode caps both by scratching and hammering on the detonating-material with pointed pieces of wood, but he never succeeded. Nor could he produce explosion by scratching with a sharp-pointed piece of iron, but hammering with the same instrument was always successful. Gently crushing the cap and the detonating-material within it will not cause explosion.

Miss-fires, whether arising from defective materials or bad workmanship, sometimes occur unknown to the miners; as, for example, when two holes are supposed to have exploded simultaneously, whereas in reality only one of them has exploded, the other having missed fire. To prevent this kind of occurrence, there should be at least two inches difference in the length of any two fuzes fired together, even though all the holes fired at that time be of the same depth.

From whatever cause, however, miss-fires arise it will probably not be possible to prevent them altogether, although they may be greatly reduced in number; for we are not likely in the future, any more than in the past, to obtain, at all times, either perfect material or faultless workmanship. The question, then, is: How can miss-fires be dealt with, so as to avoid the serious accidents which sometimes follow on them? At present the usual practice is either to remove the stemming, or by either a wooden or iron pricker to make a passage in the stemming and pass through it another cap. Both these practices are most dangerous; the latter, however, is only so when an iron pricker is used, and this is occasionally done unknown to the managers. The principal danger in both cases is of the same kind, and arises from the possibility of coming into contact with an unexploded cap and causing it to explode by a blow, either from without or within, that is, directly on the detonating-material.

For the purpose of obviating any necessity or excuse for resorting to either of these practices, the writer some time ago carried out two sets of experiments, which showed that miss-fires may be dealt with in a most simple manner and with absolute safety. One set had reference to the quantity of stemming necessary to enable the explosive to do its work properly. Shots were fired in the presence of experienced miners, who were all in favour of heavy stemming, with various thicknesses of stemming, ranging, in the different holes, from 14 inches down to $\frac{1}{2}$ inch, and these miners all agreed that the burden in each case was as well thrown as it would have been had the hole been stemmed up to the neck. A number of holes were also fired without any stemming whatever, and these were almost if not quite as satisfactory. The unanimous conclusion arrived at by all who witnessed these experiments was that 2 to 5 inches of stemming, according to the nature of the ground to be thrown, was as effective as 2 feet.

The other set of experiments were made for the purpose of determining the thickness of stemming through which a charge of gelatine-dynamite could be exploded by another charge placed on the top of the stemming. The *modus operandi* was this. Two cartridges, without a cap or fuze, were put into the bottom of the hole. These were stemmed with a certain thickness of clayey stemming; another cartridge or half-cartridge with a cap in it, and fuze attached, was placed on the top of this stemming and a further lot of stemming, generally not more than an inch or two, was put over this last cartridge or half-cartridge. The primary stemming, that between the two lots of explosive, was tried of all thicknesses up to 14 inches, and in every case the whole of the dynamite in the hole was exploded, and the

work intended to be done by it executed satisfactorily. In one experiment 5 inches of paper was put between the two lots of explosive and 2 inches of paper on the top of the last cartridge, no clay stemming whatever being used. The hole was a very strong one, in the middle of a narrow forehead, but the ground got by it was all that could have been expected if any amount or quality of stemming had been employed.

The lesson to be learned from these experiments is this: it is quite unnecessary to put more than 3 to 6 inches of stemming at the most into a hole, but if this were increased, say, to 8 inches in deep holes, so as to partly meet the prejudice of miners generally, yet even then in case of a miss-fire, all that has to be done is to put another cartridge, or part of one, on to the top of the stemming, and the first charge will be exploded in the manner described above. No unramming of a hole is necessary, and no pricker need be used, so that all the risks attending these two operations will be avoided.

The writer has caused the holes made in a number of mines to be charged as above suggested, for several months, with complete success, miss-fires being now most simply and easily dealt with.

The PRESIDENT said that the writer of the paper classed miss-fires under defective materials and bad workmanship. The last miss-fire that he (the President) saw was due to damp fuze; he did not know whether this could be called defective material. The mode of firing by introducing the exploder upon the top of the clay was rather a novelty, and he was not before aware that this method was effective.

Mr. J. L. HEDLEY (H.M. Inspector of Mines, Newcastle-upon-Tyne) said that the interesting paper read by Mr. Kendall was one which would command the attention of all connected with mining, more especially in this district where such large quantities of high explosives are used, and he did not think that any more fitting opportunity could have been chosen for the discussion of the subject. Important as many of the papers submitted to this Institute may be, it must not be forgotten that the primary object of the Institute was the prevention of accidents, and considering the large number of accidents from the use of explosives occurring annually, resulting in loss of life and serious injuries to so many people, it appeared to him that the subject of this paper was one that deserved the fullest consideration, more especially as it had been introduced by a member who had had a long connexion with the mining industry of this county, and who was, from his practical acquaintance with the working of mines, fully

qualified to express an opinion on the subject. Mr. Kendall had further supplemented this knowledge by practical experiments, the results of which it is his intention to embody in a code of special rules which, it is hoped, may be of use in the future in protecting the men against themselves, and be the means of reducing the number of accidents that occur only too frequently from the use of explosives. He would like to call for further information on one or two points:—Did Mr. Kendall mean that the exudation of the nitro-glycerine occurred in the stores before it was used, or from a delay after it was put in the shot-hole? Accidents had frequently occurred from both causes, and if any suggestion could be made for preventing such exudation it would render the use of such explosives very much safer. Should not miss-fires, due to the use of frozen material, be included under this head? It is quite true that shots miss fire from inferior detonators, but would it not be advisable to use a galvanometer and test each detonator before it is sent into the mine? And this leads up to the next point, as to defective fuze—Mr. Kendall admits that this is the most abundant source of miss-fires. Why not then take the bull by the horns, do away with fuze altogether, and use the magneto-electric battery? He had no doubt that his suggestion might sound rather revolutionary to many engineers, but if once adopted he felt sure that the change would never be regretted, and that the evils and dangers arising from the present system of shot-firing would, to a great extent, disappear, more especially if the mode of stemming the shots as advocated by Mr. Kendall (which was in force in several parts of the country) were adopted. If the battery were used the points raised by Mr. Kendall as to defective workmanship would not arise, and the temptation to unstem a shot which, if charged with any other explosive than gunpowder, is not at present prohibited by the Metalliferous Mines Act would be much minimized. Accidents from the unstemming of shots, and others caused by driving a pricker into the charge of explosive, have occurred too often, and although not contrary to the wording of the Act such practices are directly opposed to its spirit. It was satisfactory to learn that several gentlemen connected with mining in this district had taken up the matter in a way which would not only conduce to extra safety at their own mines, but which would, he trusted, be the means of inducing others to take similar steps for the protection of their workmen. In conclusion, he asked Mr. Kendall whether they were to understand that he had proved by actual experiment that a charge of dynamite could be exploded simply with the cap placed on the top of the charge?

Mr. J. D. Kendall answered yes.

Mr. R. P. W. Oswald (H.M. Inspector of Mines, Whitehaven) said that he had investigated the greater portion of the accidents occurring in this district during the last nine years, and in nearly every case he found that they had been caused by the workmen themselves, either driving the pricker, etc., into the charge or by going back too soon. At some mines there was a rule that the workmen had to wait twenty minutes in case of a miss-fire, but at many mines there was no such rule.

Mr. G. Scoular (Cleator Moor) said it had been the custom among miners generally to do nothing more than unram a small portion of the stemming. With respect to the suggestion as to electrical fuzes, this was not an infallible cure. He had been in the habit of using them—but still miss-fires had occurred, although probably bad material was the cause.

Mr. T. E. Forster (Newcastle-upon-Tyne) asked whether in the matter of putting in a charge of explosives on top of a miss-fire Mr. Kendall had experimented with different kinds of explosives, or only with the one used at the particular mines he was engaged at? He could quite endorse Mr. Scoular's remark from his own experience that electric fuzes were not free from complaint.

Mr. J. L. Hedley said he had also made similar experiments to those made by Mr. Kendall, not for any particular object, but with different explosives, and he had got the same results. A shot put in on the top of the stemming, after a miss-fire detonated both charges. The principal object, he thought, which Mr. Kendall had in limiting the amount of stemming that he put on a shot was to obviate the necessity of taking out the stemming, and of avoiding the danger and temptation to the workmen of unstemming the shot. He thought there was great benefit to be derived from using the explosive in the way suggested by Mr. Kendall. The work was done equally as well as if the shot were stemmed up to the front of the hole.

Mr. Watkyn Thomas (Cockermouth) thought that the question asked as to the kinds of explosives used was important. It occurred to one that if Mr. Kendall in his experiments got a shot to hang fire for 15 minutes and then go off, anything like twenty minutes for a miner, without a watch, to go back was cutting it too fine; he thought an hour's delay would be much safer.

Mr. J. D. Kendall thought that Mr. Watkyn Thomas was under a misapprehension as to the time during which fuze hung fire. There was no statement in the paper that he had observed any fuze hang fire for 15 minutes. He had simply made experiments as to the time

which it took the yarns to burn in the open, and he had doubled that length of time to allow for the slower combustion in the fuze. In his experience a shot would not hang fire for 15 minutes. They all knew that when waiting for a shot to go off underground, 5 minutes seemed almost an eternity, and when the miners said they had waited an hour, the time-estimate was generally very much exaggerated. As to the exudation of the nitro-glycerine, it might have taken place before it came into the hands of the consumers ; at any rate, it was only detected after a miss-fire, and, so far as he knew, there was no reason to suppose that it had not taken place before it came into the miners' possession. He thought there were very few miss-fires now from frozen dynamite, as its nature was better understood. There were perhaps not many warming-pans used, but the men had got into the way of putting the charges into their pockets, and keeping them soft in that way. It was quite unnecessary to insert the cap within the explosive. Some miners were aware of this fact, and simply dropped the cap into the hole after they had put in the dynamite. It was absolutely necessary, however, that the shot and the cap should be in contact. He had tried the experiment of putting a thin layer of loose material between the dynamite and the cap, and explosion did not then occur.

Mr. J. L. Hedley asked if the paper was removed from the dynamite ?

Mr. J. D. Kendall replied that he did not do so ; the charge was exploded through the paper.

The President proposed a vote of thanks to Mr. Kendall for his paper. There could be no doubt that the accidents which occurred from these explosives were very terrible in their results. He had known several workmen who had been permanently blinded through going back to a shot before it had exploded ; even the loss of life itself was scarcely more terrible than this ; and they welcomed any information which would help them to mitigate the danger.

Mr. T. E. Forster seconded the vote of thanks, which was heartily approved.

Mr. G. B. Forster, Past-President, took the chair, as the President was compelled to leave at this stage.

Mr. R. W. Moore's " Historical Sketch of the Whitehaven Collieries " was taken as read, and a vote of thanks was accorded to the author.

HISTORICAL SKETCH OF THE WHITEHAVEN COLLIERIES.

By R. W. MOORE.

Descriptive accounts of the Whitehaven collieries have been published from time to time, but no paper dealing with the history of the Whitehaven coal industry has yet appeared in the *Transactions* of this Institute. To members resident in, or connected with, this district, there may be little or nothing new in the account that follows ; but perhaps others not so well informed may find something to interest them in this historical description of one of the oldest collieries, and certainly the most extensively worked under the sea, in the kingdom.

According to Chancellor Ferguson, Carlisle, coal was worked in East Cumberland more or less rudely from the time of the Romans, and long before the coal-fields in the west of the county were discovered, or, if known, worked.

Although probably not worked at so remote a period as in East Cumberland, coal had been found and worked to a small extent at an early date near Whitehaven.

In 1560, it is recorded that Sir Thomas Chaloner, the lord of the manor of St. Bees, in which Whitehaven is situate, in granting certain leases of lands within his lordship, reserved to himself the right to dig for coals and other minerals, at the same time granting his tenants liberty to take for their own use at the lord's coal-pits, by the delivery of the lord's officer, as they had been accustomed to have for their own use, "so always," to quote the old document, "that they pay therefor and labour from time to time at the lord's coal-pits according as they ought to do by the custom of the said lordship." But it was not until the manor of St. Bees came into the possession of the Lowther family that coal was actively worked in it.

After the dissolution of the monasteries, the manor of St. Bees remained in the possession of the Crown till 1553, when King Edward VI. granted the priory of St. Bees, with the manor and rectory, to Sir Thomas Chaloner. It was sold by his son to Thomas Wybergh. The late Mr. W. Jackson, in a paper on "Whitehaven and its old Church," states that Thomas Wybergh seems to have mortgaged the manor of St. Bees in the year 1600 to George Lowther, the eighth son of Sir Christopher Lowther

of Lowther. Mr. Jackson further says :—"Gerard, the second son, then took an interest in it, and these dying issueless, the eldest brother, Sir John, became the possessor and transferred his title to his second son, Christopher, who was the founder of the Whitehaven branch."

It will not be out of place to refer at this stage to the history of the St. Bees grammar school royalty, which has always been worked in connexion with the Whitehaven collieries.

On August 24th, 1561, Sir Thomas Chaloner sold to Queen Elizabeth forty-eight messuages in Kirby Beacock, *alias* St. Bees, and four in Hensingham and Wray, with certain rents, all parcel of the possessions of the manor of St. Bees, the whole being of the annual value of about £21; and, on October 15th, in the same year, he farther granted to the Queen in discharge of a debt of £320 to the Crown, sixteen messuages in Sandwith, the sheep pasture called Sandwith Marsh, a tenement called Seacote, and some property in Yorkshire, producing a rental of nearly £16. All the lands thus acquired by the Crown in St. Bees and Sandwith were granted by letters patent of King James I., in 1604, to the governors of St. Bees grammar school; and the grant was confirmed by Parliament in 1606.

In the leases of 1608-9 the tenants covenant that they will permit the governors and their successors to sink pits and dig for coals in the demised premises; but there is no record of any advantage having been taken of this covenant until 1650, when the governors demised "all that pit of Coalgrove or Bearmouth, already sunk within the closes called Stephen Ridding, in the parish of St. Bees, holden of the school under the rent of 4s.," with liberty to dig for coal therein, for the term of four years at the yearly rent of £3.

In 1664, the governors granted to another set of lessees a lease of the same coal-pit for a term of seven years at the same rent. In 1679 these tenants quitted this "coalery," the coal having given out.

No profit seems to have been made of the coal under the school lands from 1680 until the year 1742, when the governors demised to Sir James Lowther all their coal-mines for eight hundred and sixty-seven years at the yearly rent of £3 10s. Since then until 1888, the St. Bees school royalty has been worked by the Lowthers; and since 1888, by the Whitehaven Colliery Company as sub-lessees.

In 1819, this lease of 1742 formed the subject of an enquiry by the Charity Commissioners; and in 1821 the Attorney-General, in accordance with the report of the Commissioners, instituted proceedings in Chancery with a view of setting aside the lease, on the plea that Sir James

Lowther and his steward, Mr. John Spedding, being governors of the school when the lease was granted, the same was therefore void. Mr. Matthias Dunn, afterwards the first Inspector of Mines for this district, was instructed, in the year 1823, to examine into the whole subject; and he narrates all the circumstances of the case in his treatise on *The Winning and Working of Collieries*.* After the lapse of six years Lord Chancellor Lyndhurst, in 1827, gave judgment. The lease was annulled, and the Earl of Lonsdale, the successor of the original lessee, was ordered to pay into court, for the benefit of the school, the sum of £5,000 as royalty on the coal obtained from the date of the lease to the commencement of the action. A new lease was directed to be granted to the earl for a term of forty-two years from January 1st, 1827, on terms to be settled by disinterested viewers. On the rates so settled, over and above the sum of £5,000 before-mentioned, royalty had to be paid on the coal wrought from the commencement of the suit until January 1st, 1827. The late Mr. Jackson states :—"On the 9th of July, 1830, it was officially reported that duly qualified persons had ascertained that £7,522 9s. 5d. was due to the governors at the said rates; and later another report was presented, certifying that £758 4s. was due to them for way-leave on the earl's coal (from Croft pit) passing through their royalty, making for the school in the whole, £13,280 13s. 5d., out of which the expenses of the suit had to be defrayed."

It is now necessary to revert to Sir Christopher Lowther, the founder of the Whitehaven branch of the family, and lord of the manor of St. Bees. Although it is not known that he took any decided steps towards opening out his coal property, it is quite certain that coal was both worked and exported to a limited extent at Whitehaven during his life-time. A print is extant depicting "The South-East Prospect of Whitehaven in the year 1642," from which it will be seen that Whitehaven was, at that date, a mere village, consisting apparently of some forty or fifty houses with a little chapel. Eight vessels are shown sheltering in a harbour that had been formed by the running out of a short pier or mole from the south side of the creek, on the site of the present old quay. The feature in the print to which most interest is attached is the train of horses, shown in the foreground, carrying packs or bags on their backs, and wending their way past the old Flatt Hall, in the direction of the harbour. It is the general opinion that these bags contained coal; but the late Mr. Isaac Fletcher, M.P., in his paper on "The Archæology of the West Cumberland Coal Trade," thinks that these pack-horses were

* Second edition, page 269.

carrying some other merchandise to the town or harbour. The writer is of opinion that these horses were bringing coal that had been wrought at Stephen Ridding, from the outcrop of the main band and six-quarters coal at Priestgill, near Keekle Beck.

Sir Christopher Lowther died in 1644, and was succeeded in the estates and baronetcy by his infant son, Sir John, who was baptized at St. Bees church in 1642. On reaching manhood, Sir John Lowther began to take the greatest possible personal interest in the extension of the trade of Whitehaven and the development of his coal-mines. Indeed he may be truly described as the founder of the Whitehaven collieries, which have ever since his day been worked with great enterprise by the noble house of Lowther until August 11th, 1888, when the working pits and a large tract of submarine coal were leased by the present Earl of Lonsdale to the Whitehaven Colliery Company, the present lessees of the Whitehaven collieries.

Perhaps the first action of Sir John Lowther, in carrying out his project of working his coal at Whitehaven, was to acquire, as far as possible, all the land round about the scene of his intended operations. In 1666 he obtained from King Charles II. a grant of such lands as had belonged to the dissolved monastery of St. Bees, but which still continued in the possession of the Crown; and in 1678 he procured a farther grant of the land between high and low-water marks. These matters being settled, for a rival but unsuccessful claim had been set up by the Earl of Carlingford and others, Sir John proceeded to carry out the ambition of his life, and so successful was he in his efforts that, in 1693, Whitehaven had grown into a thriving town with a population of 2,222 inhabitants.

Before entering upon a description of the various means employed by Sir John and his successors to develop the Whitehaven collieries, it is absolutely essential, in order that the following remarks may be readily understood, that some account, however meagre, should at this point be given of the general features of the area that has been worked over, and of the seams that have been proved and wrought.

Approximately the workings, past and present, may be taken to extend from Lowca Beck on the north to Tarnflat Hall on the south, and from the Bateman fault near Keekle Beck on the east to $2\frac{1}{2}$ miles seaward on the west.

At an early date the pits on the north-east side of Whitehaven were styled the Whingill collieries, and those on the south-west side the Howgill collieries. In fact the valley which lies between Whitehaven and St. Bees divides them.

The land on each side of this valley rises to an altitude of several hundred feet. That on the west side is known as Preston Isle, and is thought by some to have been an island at some remote period.

The coal-seams dip to the west or seaward about 1 in 12. Several seams outcrop from Ginns to Greenbank, along the west side of the St. Bees valley; and again in the vicinity of Priestgill near Keekle Beck, by reason of the occurrence of several faults, downcast to the east.

The present workings may be said to be almost wholly confined to the undersea coal. It must not, however, be supposed that all the land coal has been exhausted. Large areas of solid coal and of coal standing in pillars still remain to be worked.

The pillars in the Whingill side were left, because at the time they were formed it was thought impracticable to work them. In 1812, Mr. Buddle suggested in a report that the experiment of working these pillars should be tried. Excepting a few taken out in the main band near Windsor high, and subsequently the pillars in the same seam in the Russia district of William pit, under Bransty, Mr. Buddle's suggestion does not appear to have been followed, and the Whingill pillars still remain to be worked.

On the Howgill side the pillars were undoubtedly left for the support of the levels or water-courses, to which reference will be made hereafter.

At the Croft pit, the most southerly of the present working pits, part of the Permian series is found to overlie the Coal-measures. The pit is, as a matter of fact, sunk through the Magnesian Limestone, which is about 20 feet thick, and an underlying *breccia* only a few feet thick. Immediately to the south, these members of the Permian group dip under red shales or marls (containing beds of gypsum) which, in their turn, dip under the great mass of Permian sandstone that forms the promontory of St. Bees Head. The St. Bees grammar school coal and that lying to the south of it are covered by a great thickness of Permian strata.

The following account of the principal seams passed through in the King pit, on the Howgill side, said to have been the deepest pit in the kingdom in 1793, will probably suffice to convey a general idea of the depths and thickness of the various seams met with in the Whitehaven collieries :—

Name of Coal-seam.	Thickness. Ft.	Ins.	Depth from Surface. Feet.
Crow coal	2	2	... 180
Yard band (upper or Preston Isle)	3	0	... 480
Burnt or bannock band (including metals)...	8	0	... 600
Prior or main band	10	0	... 720
Yard band (lower)	2	8	... 774
Six-quarters or low bottom coal	5	0	... 960

With the exception of a borehole put down 360 feet below the six-quarters seam, near the Wellington pit, in 1866, nothing is known of the strata beneath that seam at Whitehaven. The borehole proved at 288 feet a seam supposed to be the Harrington four feet, and if that surmise should prove to be correct, the Carboniferous Limestone may be found at a farther depth of 354 feet below the seam last mentioned.

After this general description of the physical and geological aspects of the Whitehaven collieries, necessarily brief, because a more extended one would be quite foreign to the subject of this paper, a return must be made to the historical sketch which was interrupted at the accession of Sir John Lowther to the Whitehaven estates.

It is almost certain that Sir John did not give any serious attention to the working of his mines until the attainment of his majority in 1663, and that the first attempt at systematic mining was made by him at Whitehaven on the Howgill side, near Greenbank, at the outcrop of the prior band (as the main band was then termed, probably so called from the fact that the seam was found on lands formerly belonging to the prior of St. Bees). There is in existence at Whitehaven castle the wages book of the old Greenbank colliery. It dates back to the beginning of the year 1675, but the colliery might have been working even before then. The following is "An Accompt of all wages and disbursements upon Greenbank colliery in ye week ended ye 14th April, 1675":—

Tons.	Bur.		Days.	£	s.	d.
47	2	Three haggers 42 burthens, 8½d.	6	00	12	09
26	2	Two haggers	5	00	07	01
73	4	2 bearers, 7½d. per day	6	00	11	03
		2 bearers	5	00	06	03
		3 wattermen and bankman	6	00	16	03
		4 men in a dike 6 days and 2 men 7 days ...		01	05	04
		John Gilliat using a stone		00	00	06
		P[d] for one spoutch and 3 swills		..	,,	..9
		P[d] Item: meales smiths note		00	19	04
		P[d] Leadings 22 tons		01	02	00
		Sum p[d]		06	01	06

From this statement it appears that at that time five haggers were employed at Greenbank at a fixed wage of 8½d. a day. Their darg was twenty-one burthens each, and eight burthens made a ton. Four persons called bearers, conveyed the coals to the day-hole or bearmouth, at a fixed wage of 7½d. a day, and there the coals were teamed into a bank, whence

they were either carted or taken by pack-horses to the harbour at a cost of 1s. a ton. Although these so-called bearers derived their name originally from the fact that when coal was first worked from the outcrop it was borne out on the backs of men and even of women—as was the ancient custom in Scotland—this primitive mode of carriage could not have been pursued at Greenbank in 1675. In subsequent pay sheets corves are mentioned, and it is known that the Greenbank corf measured 32¾ inches long by 17 inches wide and 14½ inches deep, and contained 8,072 cubic inches, or say 2½ cwts. of coal. These corves were undoubtedly placed on ashen runners, resembling a sledge, and conveyed along the corf-way, constructed of two parallel lines of wooden rails. In addition to the haggers and bearers, ten other men appear to have been employed, so that the whole pit's crew numbered nineteen persons underground and one man above ground, or a total of twenty hands. The selling prices of coal at that period were:—Ships, 3s. per ton (including leading); country, 2s. per ton; and tenants, 1s. 4d. per ton.

Simultaneously with the working of the Greenbank colliery, coal was being got from the outcrops of the yard, the burnt, and prior bands seams, along the western side of the St. Bees valley, between Greenbank on the south and the fault, downcast north of 180 feet, that runs in a north-westerly direction from the head of the Ginns. The late Mr. Isaac Fletcher was certainly in error in assuming that "at first the bannock and yard seams were worked in the hillside adjoining the town, and now built over by the houses at Mount Pleasant and the Ginns" for the simple reason that neither seam outcrops on the hillside north of the dyke before mentioned. These seams were first worked on the north or downthrow side of this fault about the year 1747, at the Duke pit, where the yard band is 258 feet and the bannock band 366 feet from the surface.

The first workings from the outcrops of the seams would naturally be much impeded with the water that would follow in from the surface; and to get rid of this difficulty at the Howgill colliery, would be Sir John Lowther's object in beginning his first great engineering feat, the driving of a level or watercourse from Pow beck, which discharges itself into Whitehaven harbour, as well as for the purpose of draining a further extent of coal. This level would probably be driven about the year 1663, when Sir John had left college and settled down at Whitehaven. It was commenced at the old copperas works, as low down the stream as the buildings of the town would permit, and driven due west until it intersected the burnt or bannock band. This watercourse was then continued along the level course of that seam in a southerly direction, and passed on the east side

of Monkwray—from which it is distant about 450 feet. Consequently, the late Mr. Fletcher is wrong in stating that this level cut the bannock band 300 feet beyond, *i.e.*, west of Monkwray.

This level was continued still farther to the Knockmurton pit, which is situate close to the southern boundary-wall of the cemetery, a total distance of about 5,400 feet. This level, known as the bannock band surface water-level, is still in use for draining the water that gets access from the outcrop into Pow beck, and although small in size can still be travelled.

In the year 1698, the workings in the prior and burnt bands in the Howgill colliery, between the fault at the Ginns and Greenbank, had reached such an extent that there was an output of 350 tons weekly—drawn to bank at several pits by means of horse-gins. The Howgill corf measured 32¾ inches long, 17½ inches wide, 15½ inches deep, contained 8,883 cubic inches, and would carry possibly 2¾ cwts. of coals. Eight of these corves contained a ton, which was then a measure consisting of 36 Winchester or 12 Carlisle bushels, and weighed about 22 cwts. At that time the hewers were receiving 10d. a day, and the trailers, of whom there were 22, were remunerated with 8d. The total working cost was 1s. 11d. per ton, including 9d. per ton for leading in carts to the harbour; and the selling prices were 2s. 8d. per ton to the shipping and 2s. per ton to the country.

The old Greenbank colliery was still working in 1698, but on a limited scale, the production of coal being only 35 tons a week. At this time Sir John Lowther was working a small colliery called Lattra, situate between Whitehaven and Parton.

In 1699, a commencement was made in the yard band, the seam above the burnt or bannock band, probably from the outcrop; and in 1700 the coals therefrom, 12 corves to a ton, were drawn at the Knockmurton pit.

In 1700, a level was begun from Pow Beck, farther south, near Thicket farmhouse, and driven westward for the purpose of draining another tract of coal. This watercourse, at a distance of 1,200 feet from its mouth, cut the prior band, and was aired by two shallow pits, the Tyson and Murray, which were sunk upon it. The level was continued in the prior band, past the Baxter pit, until it reached the Greenbank dyke, a distance altogether of about 4,200 feet.

This level, considered the second important winning of the Whitehaven collieries, laid dry an extent of prior band, measuring 3,000 feet along the level line, and 1,200 feet to the rise. Out of this level, near the High House farm, a branch level was driven from the prior band westward through the Greenbank fault, and drained the Country pit district in the burnt and prior bands.

Soon after the year 1700, the new Ginns pit and the Stone pit were sunk at the Ginns, which was then a small hamlet, separated by green fields from the town of Whitehaven.

These pits were sunk down to the prior band, and were ultimately connected to the "end gills" of the various pits to the south in the same seam, thereby forming one continuous watercourse as far as the Fish pit, a distance of 4,200 feet. There the level passes through a fault into the bannock band, in which it pursues its way past Gameriggs to the Fox pit, finally to the level end in the Wilson pit bannock band. This level is now known as the Gameriggs or main band surface water-level, and is still in use from about the Hinde pit on the south to the Arrowthwaite pit on the north.

This was the third important winning of the Whitehaven collieries, and rendered still more important soon afterwards by the erection of a steam pumping-engine at the Ginns pit.

Besides the winding of water in hogsheads at the pits to the south, the water was raised (during Sir John Lowther's lifetime) at the Ginns pits by a so-called engine, but it was in reality a machine worked by horses. In a print of "The case of Sir John Lowther, Bart., and of the inhabitants of Whitehaven, in Cumberland: with reasons against a bill, to lay a duty on coals, for making a harbour at Parton, a small creek, within a mile of the said town," these levels and engines are referred to thus :—

"The country adjacent afforded coals sufficient for a staple export; but a great part of them were in the hands of several small freeholders, and could not be wrought without great and expensive levels, which must run through several people's lands, and draining all upon the rise, would enable such as bare none of the charge, to under-sell and ruine those who did; so that the working of them, under those circumstances, was impracticable, and they were lost, as well to the owners, as to the country, till Sir John Lowther, at his own costs, introduced the art of carrying on levels, and of working what was under level, by engines, a thing unknown in that county before."

These levels were used solely for drainage purposes, and the coals were drawn, as before stated, at numerous pits sunk within the area so drained. Many of the names of these early pits have not been preserved, and it is impossible to locate the position of some of the pits, the names of which appear in existing records. It would appear, however, that for a long time after the opening out of the Howgill colliery from Thicket on the south to Ginns on the north, the Knockmurton pit was the most important.

In 1705, the weekly output of the Howgill colliery was 450 tons, obtained chiefly from the prior band at the Murray pit. The burnt band was being worked at the Murray and Gibson pits, and a little quantity was being got from the yard band at the Corporal pit. The total number of hands employed were 93. Of these the haggers got 10d. per day, and the trailers 8d. per day.

Sir John Lowther died in 1705, leaving his estates to his second son James, and disinheriting Sir Christopher, his eldest son.

In 1718, it is said by several writers, that these horse-pumps, upon which Sir John Lowther so prided himself, were superseded at the Ginns pit by what was then styled a fire-engine, in other words a steam pumping-engine, which was reputed to be the second of its kind in England. It had been previously used for pumping water in London, and was purchased there by Mr. Lowther and sent by him by ship to Whitehaven. This engine had a copper boiler about 10 feet in diameter with a lead top, a brass cylinder 28 inches in diameter, wooden pumps 8 inches in diameter, with a brass working-barrel. It was, in fact, a Newcomen atmospheric engine. This engine proved so superior to the horse-pump that the fate of the latter was sealed at once and for ever.

The sinking of more pits on the Howgill side caused a great increase in the quantity of water to be pumped, and rendered another and more powerful engine necessary at the Ginns.

Mr. Lowther, afterwards Sir James, followed closely the example that had been set him by his father. He resided at Whitehaven and took the most lively interest in all that pertained to his collieries. It was he who opened out the Whingill colliery. The first pits were sunk near to the outcrop of the main band on the top of Harras Moor, at a height of about 450 feet above the sea. In the year 1716 the output from them was above 200 tons a week. About this time Mr. Lowther or his agents conceived the idea of draining the coal under Harras Moor by means of a level watercourse, which was commenced from Bransty Beck at the bottom of Wheelbarrow Brow, on the east side of Whitehaven. This level in course of time was driven past the Harras, Jackson, Fox, and Hunter pits through the fault (downthrow east 300 feet) which separates the Bateman pit workings from the remainder of the Whingill colliery, and terminates in the main band level end, close to the Bateman pit, a distance of about $1\frac{1}{2}$ miles.

This was the fourth great winning of the Whitehaven collieries, and drained all the Whingill colliery in the bannock and main bands, except the workings in the George, Lady, and James pits, which are below its level.

The water from the George and Lady pits was pumped at the George pit up to a level 750 feet long, by which it was discharged into the Bransty Beck, about 600 feet south of Lonsdale Place. The James pit main band was drained to the Saltom pit on the Howgill side through the single stone drift, which was driven in 1796 underneath the town of Whitehaven, through the Norway dyke (upcast to the north-east of 240 feet) and which, until a connexion was made many years afterwards seawards, was the only communication underground between the Howgill and Whingill collieries.

In 1725, Sir Christopher Lowther, who had been disinherited by his father, died and Mr Lowther succeeded to the title.

In all his undertakings at the Whitehaven collieries Sir James Lowther had the assistance of the two brothers, John and Carlisle Spedding, who succeeded their father, Edward Spedding, who had become chief steward of the Lowther estate at Whitehaven in 1685. But it was Carlisle Spedding who had the practical conduct of the collieries, and who conceived and carried out all the great schemes in Sir James Lowther's time. He was a man of considerable ability and attainments, and under his management the Whitehaven collieries were extended in every direction.

It is on record that Sir James Lowther sent Carlisle Spedding to Newcastle to learn all the improvements in coal-mining to be seen there, in order to apply them at Whitehaven. To effect this object, Mr. Spedding got himself engaged as a hewer at various collieries in that neighbourhood under a fictitious name. After being there some time, he had the misfortune to get severely burnt by an explosion of fire-damp. The extraordinary care and attention, bestowed by the best doctors whose services could be procured, upon apparently a humble collier, led to the discovery of his identity and his mission, and ultimately to his return to Whitehaven to put into practice what he had seen and learned.

Mr. Carlisle Spedding was, about 1730, the inventor of the steel-mill, the first attempt to light with safety coal-miners at their work in an atmosphere containing an admixture of fire-damp. The steel-mill was merely a small steel disc made to revolve with great rapidity, by means of a cog-wheel and pinion against a piece of flint, and the stream of sparks thereby emitted afforded the light by which the man had to work. Of course it was only resorted to in places making gas. The steel-mill was used on that account when the communication between the Howgill and Whingill collieries, above referred to, was being made; and, in a description of that work, it is stated that the mills would give no light farther than where there was atmospheric air. Mr. Carlisle Spedding is also credited with having made the first improvement in the ventilation

of coal-mines by the system known as coursing of the waste, "which consisted of threading the air-column up certain workings and down others until it ventilated the whole waste."* He also, judging from the old colliery pay sheets, introduced the use of gunpowder into the Whitehaven collieries about the year 1730, for the purpose of stone drifting which had hitherto been done by what was known as plug-and-feather.

The success that had attended the introduction of the steam engine for pumping, at the Ginns pit, water that drained from all the pits bordering upon the outcrop of the seams in the St. Bees valley, undoubtedly led Mr. Carlisle Spedding to propose to Sir James Lowther the bold project of sinking a pit close to the seashore, as far to the dip as possible, for the purpose of winning and working not only the land coal which lay between the coast-line and the workings to the rise, made from the Greenbank, Corporal, Wilson, Radcliffe Bank, Causey Head, Burnt, Swinburn, Grayson, Bell, Fox, Double, Miller, Skelton Bank, Howgill Heads, Pickeron Bank, Senhouse, Gill Close, Gibson, Murray, Knockmurton, Mawson, Pow, Andrew, Forbes, Granger, Darby, Gameriggs, and Baxter pits, but also to work the coal under the sea.

The site selected for this great undertaking was Saltom, close to high-water mark, and a commencement was made with the sinking in 1729. This was the fifth great winning of the Whitehaven collieries. The shaft was sunk oval, 10 feet by 8 feet, a shape that commended itself to Mr. Carlisle Spedding and his successor, his son James. All the pits sunk during their terms of office were of this form.

When the pit was down 252 feet from the surface, a strong blower of gas was pricked, and was piped to the pit top, where it burned for many years afterwards. This blower formed the subject of a paper communicated to the Royal Society in 1733. Fire-damp was then no new phenomenon in the Whitehaven pits, as the pay bills from the earliest times testify. There are frequent entries of items such as:—"Fireing damp," "oyle and brandy for a burnt man," paying money to "burnt men," and an annual payment of £20 to a medical man "for cureing burnt men."

Mr. Spedding offered to supply the trustees of the harbour with whatever gas they required to light the town, if they would be at the expense of conducting it through the streets. It is not certain whether or not the trustees availed themselves of this offer, but it is a fact that Mr. Carlisle Spedding conducted gas from an adjacent pit, it might be Saltom, through a leaden pipe to the laboratory of Dr. Brownrigg, an eminent scientist, who resided in Whitehaven, and who married Mr. Carlisle Spedding's niece.

* M. Dunn, *Winning and Working of Collieries*, second edition, page 151.

Saltom pit was completed to the main band, a depth of 456 feet, in the year 1731. At the pit-top, a fire-engine, with a 12 feet boiler, a cylinder 40 inches in diameter, and a pump 7 inches in diameter was erected. The pumps were in four lifts, and the engine was a Newcomen atmospheric, similar to that at the Ginns. In the course of a few years this engine was found inadequate to deal with the feeder of water, and a duplicate engine was in consequence added. In 1782, both these engines were replaced by an engine that was, at that day, regarded as a great mechanical wonder. It was an atmospheric engine with a cylinder 70 inches in diameter and 6 feet stroke. It had an air-pump 3 feet in diameter and 3 feet stroke. The beam was oak, 24 feet 3 inches long, 21 inches deep, and 19 inches broad. There were four lifts of pumps, the two top sets being 11 inches, and the two bottom sets 11⅝ inches in diameter. There were three malleable-iron boilers, each 13 feet 6 inches in diameter, 9 feet 4 inches high, with hemispherical tops. This engine continued to work until the year 1866, when it was broken up, and the water pumped at the Wellington pit. In the year 1832, the great Saltom pumping engine was supplemented by a high-pressure engine with a cylinder 2 feet 11 inches in diameter and 7 feet stroke, working 10½ inches pumps.

The coals at the Saltom pit were drawn to bank at the shore-level by gins, conveyed by a short drift to the bottom of the Ravenhill pit, and there drawn to the top of the cliff, a height of 162 feet, again by gins. Thence the coals were taken by waggons to the staith at the south side of the harbour. It is generally believed that Mr. Spedding made the first waggonway to the Parker pit; but the writer thinks that the first railway of that kind was made by him either to the Ravenhill pit, or to the Saltom pit before the Ravenhill pit was sunk. The Parker pit was not sunk until 1734, and then only to the yard band, a depth of 240 feet. The Parker pit waggon-road is shown in the "Bird's eye view of Whitehaven," engraved by Richard Parr, from a painting by Mr. Matthias Read, executed in 1738.

The waggons held 44 cwts. of coal and ran on wooden rails. On the Howgill side, these railways were laid from the pits to a large staith, capable of holding 3,000 waggons of coals, that was constructed close to the quay on the south side of the harbour. The coals could either be shipped from the five hurries which were erected there, or stored in the staith, which was for the greater part covered in. All the waggon-roads had a descent to the harbour. The full waggons were braked down by means of a convoy fixed to the side of the waggon, and the empties were taken back to the pits by horses. The empties on the Howgill side went

back by the old bowling-green near the present Red Flag Inn, and to avoid making sharp curves turntables were put in at the angles. The coals from the Whingill pits were conveyed at this time by a similar waggon-road to a large staith situate on the site of the present cab-stables, and thence they were carted to the shipping. To dispense with this dilatory and costly mode of shipment, it was decided to carry the waggon-road direct to the shipping-berths on the north wall across Tangier Street. Bransty arch was built with this object, and the incline over it was opened with great *éclat* on August 9th, 1803.

In 1731, for the week ending March 8th the output was as follows:—

HOWGILL COLLIERY.

	Tons.	Loads.	Tons.	Loads.
Corporal pit	540	0		
Swinburn pit	480	0		
Harrison pit	148	0		
Greenbank pit	37	4		
Saltom pit	296	4		
			1,502	0

WHINGILL COLLIERY.

Pearson pit	...			
Carr pit	36	0		
Pedlar pit	80	0		
Taylor pit	175	0		
			291	0
Total			1,793	0

The prior band was the only seam that was being worked during this particular week. The hewing prices varied from 2½d. per ton on the Howgill side to 1s. per ton on the Whingill side. It will be observed that the payment of haggers by day had at this time ceased. The system of payment by measure probably commenced about the year 1720, for the ton mentioned above was simply a measure and shortly afterwards and up to within quite recent times the men were paid by corf or basket.

On August 5th, 1737, a violent explosion, styled "the great damp" in the pay sheet, occurred at the Corporal pit, by which 22 men and 3 horses were killed. This is the first great explosion in the Whitehaven collieries of which there is any record.

The sinking of the Saltom pit did not complete the list of Mr. Carlisle Spedding's achievements. On the Howgill side, he sank the Thwaite, King, Duke, Kells, Fish, Newtown, Country, Moss, and Hind pits, and deepened several others. On the Whingill side he sank the Carr, Pearson, Pedlar, Taylor, Fox, Daniel, Jackson, Hunter, Watson, Harras, Green, and other pits.

In 1755, on the Howgill side, the Thwaite, Howgill Heads, and Kells pits were working in the prior band; the Saltom, King, Fish, and Fox pits in the bannock band; and the Gameriggs and Hinde pits in the yard band. The weekly output was 2,400 tons produced at a total cost of 1s. 10d. per ton. On the Whingill side, the Carr, Jackson, and Pearson pits were producing an output of some 600 tons a week at a cost of 2s. 6d. per ton.

On January 2nd, 1755, Sir James Lowther died, at the age of 81 years. He was buried at Trinity church, the only Lowther ever buried at Whitehaven. He had never been married. In his will, dated September 14th, 1754, he says, "I give unto John Spedding, my steward, £1,000 for his long and faithful services; and for the like account I give to his brother, Carlisle Spedding, £500." The population of Whitehaven had then reached 11,000 inhabitants.

Mr. Carlisle Spedding did not long survive his employer. On August 8th, 1755, he was killed down the pit by an explosion of fire-damp, at the age of 59 years. No further particulars of his death exist.

Mr. Carlisle Spedding was succeeded in the management of the Whitehaven collieries by his son, James, who, on the death of his uncle, Mr. John Spedding, was appointed steward, uniting, in his person, the offices held formerly by his father and his uncle.

Sir James Lowther was succeeded in the Whitehaven estates by Sir William Lowther, of Holker, who died in April, 1756, and was buried in Cartmell church. During his short occupation of the estates no new works were undertaken. Under the will of Sir James Lowther of Whitehaven, the Whitehaven estates then passed to Sir James Lowther of Lowther.

Mr. James Spedding still continued to be steward and colliery agent, and under his sway the Whitehaven collieries continued to prosper, and to be worked with vigour both under land and sea, under the latter from the Saltom pit.

In 1765, M. Jars, a member of the French Academy, visited the Whitehaven pits, and wrote a graphic description of them in his work entitled *Les Voyages Métallurgiques*, from which it is evident that the workings at Saltom pit were two-thirds of a mile under the Solway Firth at that time. M. Jars describes the method by which the water was brought to the shaft from the dip. There was a double line of rails laid in the dipping, and a horse-gin placed at the top pulled up the full tub on one side whilst the empty tub passed down on the other. By an automatic arrangement the tub full of water discharged itself at the top of the brow, whence the water ran to the shaft. The water-tubs were higher at one end than the other, to allow for the dip of the seam.

There is every reason to believe that the coal worked under the Solway at Saltom was the first ever worked beneath the sea in any part of the world. The sea-coal at Saltom, and afterwards at the William and Wellington pits, was worked free of royalty until the year 1860, when the Crown put forward a claim to the minerals lying below low-water mark. From 1860 until 1880, the submarine coal was worked under a Crown lease. In 1880, Henry, Earl of Lonsdale, purchased the minerals under the sea down to the bottom of the Coal-measures, and for the distance of ten miles seawards.

In 1765, the following pits were working, during the week ending on November 6th, with the following results :—

HOWGILL COLLIERY.

	Tons.	Loads.	Tons.	Loads.
Duke pit, prior band	216	0		
Kells pit „	155	1		
„ bannock band	180	0		
King pit, prior band	352	4		
„ bannock band	180	0		
Fox pit, prior band	216	0		
Wilson pit „	108	0		
Hinde pit „	195	0		
Saltom pit „	285	0		
Fish pit, bannock band	150	0		
Thwaite pit, yard band	216	0		
			2,253	5

WHINGILL COLLIERY.

Jackson pit, prior band	324	0		
Pearson pit „	180	0		
Fox pit „	225	0		
Hunter pit „	70	0		
Pedlar pit „	127	0		
Scott pit „	168	0		
			1,094	0
Total			3,347	5

The shipping price was then 3s. 4d. per ton.

The chief winning during Mr. James Spedding's stewardship was the sinking of the Croft pit. It was drawing coals from the prior band in 1774, and—with the exception of short stoppages in 1863 and 1875 for repairs—it has worked continuously ever since until the present day. It was not until 1818 that this pit was sunk to the level of the six-quarters seam, a farther distance of 270 feet.

On the Whingill side, the Wolfe, Davy, Lady, Bateman, George, North, and Howe pits were sunk to the main band, the Scott pit to the six-quarters coal, and the Moss pit (Westcroft) to the yard band, during the agency of

Mr. James Spedding, who retired in 1781 from the practical management and was succeeded by Mr. John Bateman. Mr. James Spedding died on August 22nd, 1788, in the 68th year of his age, and was buried at Trinity church, Whitehaven.

In 1781, when Mr. Bateman took office, the following pits were at work on the Whingill side, and were raising, per week, when full time was worked, the following quantities:—

	Tons.
Lady pit, prior band	216
George pit, ,, ...	198
Davy pit, ,, ...	262
North pit, ,, ...	144
Jackson pit, ,, ...	144
Pearson pit, ,, ...	210
Bateman pit, ,, ...	288
Howe pit, ,, ...	270
Total	1,732

At this time the Taylor, Hunter, Carr, Fox, Daniel, Green, Watson, Pedlar, and Harras pits, all sunk in the time of the first Sir James Lowther, had been abandoned, and the Wolfe, Scott, and Moss pits (Westcroft) had also been stopped.

At this date the following pits were being worked at the Howgill side, and attached to each are the numbers of tons, per week, capable of being turned out when full time was worked:—

	Tons.
Duke pit, prior band	216
King pit, ,, ...	360
,, bannock band	180
Kells pit, prior band	180
,, bannock band	180
Croft pit, prior band	648
,, bannock band	216
,, ,, ...	144
Total	2,124

At this time, it is stated that all the prior band was worked at the Fox pit, Howgill colliery, and that the workings in the same seam at the Wilson and Saltom pits, and on the bannock band at the Duke pit were standing.

The number of horses employed at this time in the colliery was very large. For instance, at the Howgill side, there were then employed underground 69 tram-horses, and aboveground 80 gin-horses, and 24 bank-horses; a total of 173 horses. Of this number, 56 were provided by the neighbouring farmers.

If the Whingill side required horses in the same proportion to the output as the Howgill side did, the total number employed there, both above and below ground, would probably have been about 140, making a grand total of 313 horses for all the collieries.

It was no wonder, therefore, that attention should be directed to the application of steam-power for winding coals. Indeed it is extraordinary that the steam engine should have been so long in use before any attempt was made to apply it, in place of gins, for the winding of coals out of the pits.

The first attempt in this direction was made at the George pit, in the year 1787, and could not be said to have been a great success. On the same shaft as the rope-rolls, there was an overshot waterwheel, divided by a diaphragm with buckets set the reverse way on each side. The engine pumped the water on to the top of this waterwheel into the buckets on either the one side or the other of the diaphragm, according to the way in which the rope-roll was desired to revolve. The revolution was reversed by turning the delivery of water from the pump into the contra set of buckets. The reason why the waterwheel was employed was probably because it was not then known that rectilinear could be converted into rotatory motion by means of the crank. This awkward pumping arrangement was discontinued in 1803.

At Whitehaven the first application of the steam engine, working on to the rope-roll shaft with a crank and connecting-rod off the beam, was made at the Davy pit, on the Whingill side, where an atmospheric engine was erected by Messrs. Heslop & Millward, of the Seaton Iron Works, near Workington, in 1791. There were two cylinders—one at each end of the beam, which was made of wood. The larger cylinder was called the hot cylinder, and was open-topped as in the atmospheric engine: the smaller cylinder at the other end of the beam, termed the cold cylinder, was fixed in a cistern containing water. The steam from the hot cylinder was condensed in the cold cylinder by water, which was forced through it into the cistern by means of a small pump worked off the beam. This engine was afterwards removed to the North pit. Similar engines were erected at the Kells pit, Howgill colliery, in 1793, and at the Lady pit, Whingill colliery, in 1795. This type of engine was patented in 1790 by Mr. Adam Heslop, of Ketley, in the County of Salop, and was extensively used at other collieries in West Cumberland.

The Kells pit winding-engine was removed to the Wreah pit, near Hensingham, and worked there until May 25th, 1878, when the pit was abandoned. It was then placed in the Science and Art Museum, at South Kensington, where it may now be seen.

In 1784, Sir James Lowther was created first Earl of Lonsdale.

On January 31st, 1791, an event occurred which created a great sensation in Whitehaven. The main band, over the Norway fault, was being worked underneath the town, and the workings had then reached the neighbourhood of Duke Street, when the rise crosscuts holed into an old waste, of which there was and is no account, and liberated a large quantity of water, drowning two men, one woman, and five horses in the workings. A model of these workings may be seen at the Granary Yard, Whitehaven. The ground suddenly shrank in the garden behind Somerset House (then the residence of Mr. H. Littledale) which, together with a number of other houses in Scotch Street and George Street, were cracked and otherwise damaged. The inhabitants in this locality were thoroughly alarmed, deserted their dwellings, and camped out with their furniture in the streets until they were assured by competent viewers, who were brought to the town and inspected the workings, that all danger was over. The water ran off in a few hours. There can be no doubt that an old waste under the White Park had been tapped. The seam at the lower end of the Park lies at a shallow depth, and must have been wrought in very ancient times. Immediately the water had run off, dams were placed in the crosscuts, and have effectually prevented any further influx.

In 1794, Whitehaven had so flourished that it possessed a population of 16,400 inhabitants.

In 1797, Lord Lonsdale was created Viscount Lowther of Whitehaven, with remainder to the heirs male of the Rev. Sir William Lowther, Bart., of Swillington. On May 24th, 1802, the Earl of Lonsdale died without issue, and was succeeded in the title of Viscount Lowther of Whitehaven by Sir William Lowther, eldest son of the Rev. Sir William Lowther, of Swillington, to whom he bequeathed almost the whole of his princely fortune.

In 1800, the James pit, Whingill colliery, was sunk; but the great work of which Mr. Bateman was the author was the sinking of the William pit, on the shore near Bransty, on the north side of the harbour. This may be termed the sixth great winning of the Whitehaven collieries. It was begun in the year 1804, but was not fully completed till 1812. It reached, at the end of the year 1805, the main band, which it had then become the custom to call the prior band. From that date until the present time, the William pit has continued at work in the main and bannock bands, and the workings now extend for many miles seaward. The first winding-engine or rotation machine, as it was then called, at this pit was erected in 1806. It was a Heslop engine, with hot and cold

cylinders, 44 inches and 28 inches respectively in diameter, and 5 feet 6 inches stroke. It was supplied with steam from two malleable-iron boilers 13 feet 6 inches in diameter, 9 feet 4 inches high, with hemispherical tops. This engine was pulled down in 1850, and replaced by a high-pressure one. In 1810, a fine atmospheric pumping-engine was erected. It had a cylinder 82 inches in diameter, and the stroke was 8 feet. The original beam was composed of wood, but was replaced by one of iron with parallel motions at each end. The air-pump was 3 feet in diameter, and 3 feet 8 inches stroke. The pumps were 12 inches in diameter, and the height that the water was lifted was 654 feet. The old engine is still working.

The first coal shipped from the William pit was on March 10th, 1806.

In 1804, a winding-engine with a conical drum was erected at the North pit.

In the year 1807, Viscount Lowther was created Earl of Lonsdale (the second creation).

In 1811, Mr. Bateman retired, and was succeeded by Mr. John Peile, who had been trained from his youth in the Whitehaven collieries. Mr. Bateman died in 1816, in the 67th year of his age, and was buried in Trinity church, Whitehaven.

In 1811, the pits working during the week ending January 5th were :—*Howgill colliery*—The Saltom, Moss, Kells, Croft, and Wilson pits producing 2,062 tons of coal. *Whingill colliery*—The William and North pits producing 1,440 tons of coal.

The selling price of coal at the staiths was then 4s. 4d. per ton. It was about this time that the wooden waggonways were relaid with cast-iron fish-bellied rails.

In 1812, Mr. Taylor Swainson, engineer for the Whitehaven collieries, tried his invention of the "iron horse," or locomotive engine, on the colliery waggon-roads. The engine answered admirably, but its use had to be discontinued, owing to the track not being strong enough to bear it. It has been said that Mr. Taylor Swainson was the real inventor of the locomotive.

Under Mr. Peile's direction, the Whitehaven collieries made great progress.

In 1813 he constructed the Howgill surface-incline down to the Howgill staith. In 1818, he sank the Croft pit from the main band down to the six-quarters seam, and drove therefrom a level drift until it cut the main band seaward. In 1819, he did a similar thing at the Saltom pit. He sank a new shaft at the Duke pit from the surface to the six-quarters, and also the James pit from the main band to the six-quarters.

In 1832, he began the Parton drifts and the Countess pit. The Parton drift was intended, like the Whingill levels, to drain a vast tract of inland coal; but, although it has been driven for a distance of about $1\frac{1}{2}$ miles, it has failed to win such an extent of coal as was done by this means under Harras moor. Nevertheless, it may be termed the seventh great winning of the Whitehaven collieries. The Countess shaft furnishes a splendid example of pit-walling; and it is also remarkable for having two stone brattices built at right angles to each other, and dividing the shaft into four compartments. Each course of the brattice is arched and let into the side of the pit. Thus a hole could be knocked through the brattice at any point in the shaft, without weakening the part above. The Countess pit worked until 1863, when it was abandoned, and when also the Parton drifts were stopped. The Wreah pit was also sunk during Mr. Peile's tenure of office.

The undertaking with which Mr. Peile's name will always be associated is the sinking of the Wellington pits, which were begun in 1840. In 1843, one of the pits was down to the main band, and the other to the bannock band; both reached the six-quarters coal in 1845, and thence level stone-drifts were driven seaward and intersected the main band at a distance of 2,700 feet. The Wellington pits may be fairly denominated the eighth great winning of the Whitehaven collieries. Mr. Peile proposed to sink the pits to a farther depth of 960 feet, a total depth of 1,800 feet from the surface. The last 600 feet would have been in the Carboniferous Limestone. From the bottom of these pits, in the limestone, he proposed to drive a pair of level drifts westward until they intersected the six-quarters coal and main band, and finally the bannock band at a distance of 13,500 feet from the shafts. The first 6,000 feet of these drifts would have been in limestone.

Acting upon the instructions of Viscount Lowther, who was then taking an active part in the affairs of his father, the Earl of Lonsdale, Messrs. George Stephenson (the celebrated engineer) and Frank Forster made a report upon the Whitehaven collieries on September 9th, 1843. They approved of the site of the Wellington pits, the sinking of the same to the six-quarters seam, and the driving of a pair of drifts therefrom westwards to cut the main band underneath the sea; but they disapproved of sinking the pits to the limestone, because at such a depth "it would be expensive and inconvenient to raise the quantity required, setting aside the dangers arising from the breakage of ropes," and "notwith-

standing the compact nature of the limestone in this district," it might be "so affected by faults as to leave cavities and fissures yielding such a quantity of water as to render the colliery unworkable, such cavities and fissures being very common in the Mountain Limestone."

Mr. Peile's Wellington pit winning stopped at the six-quarters seam, and no further exploration of the strata below that seam has been made, save by the bore-hole mentioned in the earlier part of this paper.

Mr. Peile commenced the manufacture of coke and patent fuel at both the William and Wellington pits, but there was no demand at that time for those products, and, during the term of office of his successor, these branches of industry ceased. He also, in 1839, introduced the use of screens at the Whitehaven collieries.

In 1841, a fan was erected at the upcast at Duke pit for the ventilation of the workings in the main band and six-quarters seam at that pit. The fan was 8 feet in diameter and 4 feet wide. It was driven by a high-pressure engine with a cylinder $12\frac{3}{8}$ inches in diameter by 3 feet stroke, and produced 23,000 cubic feet of air per minute. This was the first attempt at mechanical ventilation at the Whitehaven pits, but it could not be considered a great success; and the collieries continued to be ventilated by underground furnaces, as hitherto, until 1870, when a Guibal ventilator was erected at the Duke pit on the advice of Mr. G. B. Forster.

During Mr. Peile's reign at the Whitehaven collieries the output reached about 250,000 tons a year. In 1818, the average selling price of coal was 9s. per ton, but in 1843 it was only 6s. per ton, 7s. 6d. per ton, excluding small coal.

On March 19th, 1844, William, the second Earl of Lonsdale, died in the eighty-seventh year of his age, and was succeeded by his eldest son William, Viscount Lowther, who had for some time past taken his father's place in looking after the Whitehaven colliery and estates.

In 1847, the Wilson pit, and in 1848, the Saltom pit, ceased drawing coals.

In 1848, Mr. John Peile retired, after long and faithful service. He was succeeded in the management of the collieries by Mr. Peter Bourne, who resided at Somerset House, which had been purchased by Lord Lonsdale after the damage that had been done to it in 1791. Mr. William Anderson, of South Shields, was retained as consulting viewer.

On January 17th, 1855, after a few days' illness, Mr. Peile died in the seventy-ninth year of his age. He was interred in St. James' churchyard.

Excepting perhaps Mr. Carlisle Spedding, he was without doubt the ablest viewer and citizen that Whitehaven has produced. It its obituary notice, the *Cumberland Pacquet* said of him:—

In early life Mr. Peile entered the service of the Earl of Lonsdale, and being a man of great perseverance, he soon attained distinction in his situation, and ultimately succeeded to the principal agency of his lordship's extensive collieries in this neighbourhood, a situation which he filled in a manner highly creditable to himself, with great advantage—and, we believe we may add, with perfect satisfaction—to their noble owner, during a series of nearly forty years. Throughout the whole of that lengthened period we may safely state that Mr. Peile enjoyed the implicit and unbounded confidence of his employer, and never, we may with equal certainty add, was confidence more worthily bestowed or duties more zealously and faithfully discharged. For upwards of half a century Mr. Peile took a prominent part in all public matters connected with the port, evincing on all occasions an ardent desire to benefit and improve his native town, and to whom Whitehaven is more indebted for the great improvements which have been effected within that period than to any other individual. In fact, his great object, as manifested throughout a long and useful life, was to advance the prosperity of the port, to suggest and carry out such undertakings as might tend to that end, and at the same time be a lasting ornament to the place. It would perhaps be more difficult to point out any important work, either in connexion with the town or harbour, effected within the period indicated, in which Mr. Peile was not prominently concerned, than to enumerate a series of the great and durable undertakings in which he was the primary mover, and which were essentially benefited by his matured judgment and good taste. Some of the most important of these we briefly enumerate. Mr. Peile was a member of the Board of Trustees of the town and harbour for upwards of forty years, and during three years previous to his retirement from public life he was chairman of that body. He took a warm interest in the erection of our magnificent piers, which have perhaps no equal in the kingdom. We are also indebted to him for our present market house (since replaced)—at all events the project originated with him, and by his influence and perseverance the work was accomplished. The same may be said of the Public Library, Catherine Street, the principal room in which was finished, after a handsome design, by the subject of this brief notice. Having thus wandered into Lowther Street we cannot pass unnoticed the Joint Stock Bank and the Bank of Savings, the erection of both of which originated with Mr. Peile. In the latter establishment he was deeply interested, and of the other he was president. Though these handsome structures may perhaps be regarded as comparatively private property, we point them out as evincing his good taste and his desire to improve the appearance of the town. Mr. Peile was one of the trustees of the Whitehaven turnpike roads, and in that capacity he displayed the same sound judgment as in other multifarious matters which came under his notice. In him all the great improvements on the trust road had a zealous and judicious advocate. He always took a lively interest in the prosperity of the Infirmary, of which institution he was the treasurer from its establishment up to a very few years before his death. Many important works connected with the collieries were completed under his directions whilst in office, one of which claims a passing notice from its magnitude and the manner in which it was accomplished. We hardly need say that we allude to the Wellington pit, which for convenience and finish has perhaps no equal in the kingdom, and which for ages to come will be regarded as a monument of his judgment and taste. The important situation he filled necessarily brought him in contact with opposing political parties, yet to what-

ever height antagonism of that nature might rise, he never suffered such difference on public grounds to mar the intercourse of private life. On this point it has been justly observed of the deceased gentleman that he respected his opponents, and was ever ready the day after the battle to meet them with the full cordiality and candour of his nature.

A most important matter in connexion with the development of the collieries occurred whilst Mr. Bourne was agent. About the year 1849, the workings in the Croft pit main band had reached the downthrow fault, known as the St. Bees dyke, that had already stopped the workings in that seam, in the Wilson pit to the south. It was thought by many that the southern extremity of the West Cumberland coal-field had been reached, and doubtless they were induced to form this opinion from the fact that the Permian sandstone puts in an appearance above the Coal-measures not far off. A level drift had been driven through the fault for a distance of upwards of 900 feet, and several boreholes had been put—both up and down. Eventually in 1862, a borehole at the end of the drift in the school royalty proved the main band at a depth of nearly 318 feet. So important was this discovery considered that the journal of this borehole was engrossed, in duplicate, on parchment, and signed by Mr. Stephen Toward, the borer; Mr. W. Anderson, the consulting viewer; Mr. Peter Bourne, the colliery agent; Mr. H. Mulcaster, viewer; and Mr. John Ellwood, the overman of the pit. This fault was afterwards crossed seaward, where the throw was found to be much less, and the Croft pit workings now extend nearly a mile beyond it.

Nothing else worthy of remark occurred during Mr. Bourne's term of office except the fire that broke out in the six-quarters seam at the Wellington pit on August 24th, 1863. At that time there were two underground hauling-engines, with boilers, placed near to the level-drifts, to which allusion has been made, leading from the six-quarters seam to the main band. The smoke from the boiler-fires was conveyed by means of a chimney-drift to the Duke pit. This underground flue passed through the six-quarters coal, near the Wellington pit, and it was there that the fire was discovered. It was thought by some that the fire originated in the chimney-drift from the ignition of some soot by a spark, and that the fire thus created had penetrated through some crack in the arching to the coal. Others thought that the fire was the work of an incendiary. In this emergency, the late Mr. T. E. Forster, of Newcastle-on-Tyne, was telegraphed for, Mr. Anderson, Lord Lonsdale's consulting viewer, having died in 1862. It was upon Mr. Forster's advice that this terrible underground fire was extinguished. How this was accomplished is too long a story to tell here in detail, although worthy of more than a passing notice.

The means adopted may be briefly described as the sealing hermetically of the Duke pit, and the flooding of the entrance to the six-quarters seam at the Wellington pit-bottom.

This latter operation was effected by putting in four dams, two in the drifts leading through to the Saltom pit, and two in the drifts leading from the six-quarters seam, at the Wellington pit, to the main band seaward, and by pouring in the sea-water down one of the Wellington pits by means of a drift, fitted with sluices, that was driven from shore between high and low-water marks. In addition, steam was blown through a pipe in the tight stage at one of the Duke pits. Although several attempts were made to open out the six-quarters district, where the conflagration was, it was not until 1866 that the fire was completely extinguished. It is worthy of note that this fire, which at one time threatened the safety of the whole colliery, was put out without loss of life, and, with one or two minor exceptions, without accident or injury to any one. But the loss occasioned by the almost total stoppage of the Wellington pit was great. Besides the loss of the six-quarters district, the workings to the dip, below the spending of the drifts seaward to the main band, were flooded owing to the leakage of the dams beforementioned. Work was found for a limited number of men in the main band pillars in the King pit district, but it was not until 1872 that the water was got out of the dip districts so as to enable work to be resumed there.

The pits working at the Whitehaven collieries at the time of the Wellington pit fire were the Wellington and Croft pits on the Howgill side, and the William and Wreah pits on the Whingill side. The Countess pit had just been abandoned.

After the death of Mr. Anderson, Messrs. T. E. Forster, G. B. Forster, and T. G. Hurst, of Newcastle-upon-Tyne, were appointed consulting engineers, and, on the resignation of Mr. Bourne in 1867, they took the direct management of the Whitehaven pits into their hands, having first of all Mr. Henry Mulcaster as resident viewer, and after his death Mr. Matthew Harper. During their term of office many improvements were made. The air-courses were greatly improved, mechanical ventilation introduced, new haulage-roads made, and new hauling-engines erected at the pit tops.

In 1866, the present pumping plant at the Wellington pit was erected. The engine, which is single-acting high-pressure, has a steam cylinder, 90 inches in diameter, and 10 feet stroke. Thore are three sets of pumps, viz.:—Top lift, ram, 20 inches in diameter, 270 feet ; middle lift, ram,

20 inches in diameter, 270 feet; and bottom lift, bucket, 20½ inches in diameter, 300 feet.

In 1870, the sinking of the Henry pit was begun, and shortly afterwards the William pit was put down to the six-quarters seam from the main band.

On March 4th, 1872, William, Earl of Lonsdale, died, aged eighty-four years, and was succeeded by his nephew, Colonel Henry Lowther, in the estates and title.

It was during the lifetime of Henry, Earl of Lonsdale, 1874, that, in accordance with the advice of Mr. G. B. Forster, of Newcastle-on-Tyne, his lordship's consulting engineer, a borehole was put down at Gutterfoot, near St. Bees, by Mr. Vivian, with the diamond rock-drill, and proved the main band seam at a depth of 1,440 feet from the surface. This discovery is of vast importance. It proves the continuity of the coal from the Croft pit workings, which are about two miles off, and is a certain indication that an immense virgin coal-field lies both inland and seaward to the south.

Here the writer will conclude, the historical sketch having been brought down to a quite recent date, because the Whitehaven collieries of recent years will be described by Mr. H. M. James, and the coal-field of South Cumberland will form the subject of a paper by Mr. R. Russell.

Finally, the writer acknowledges the valuable assistance Mr. T. Studholme and his brother, Mr. W. Moore, have rendered him in the preparation of this paper.

The PRESIDENT read a paper by Mr. R. Russell on "The Extension of the West Cumberland Coal-field Southward and Northward under the St. Bees Sandstone," which will be published in a succeeding issue of the *Transactions*.

DISCUSSION ON MR. J. P. KIRKUP'S PAPER ON "THE SINGARENI COAL-FIELD, HYDERABAD, INDIA."*

The HYDERABAD DECCAN COMPANY LIMITED (London) wrote that the attention of the Board of Directors had been called to this paper read before the Institute, and they make the following communication :— The Singareni coal-field is held under a long lease by this company, and Mr. Kirkup was employed as colliery manager under an agreement for three years, which expired on August 15th, 1892. The directors

* *Trans. Fed. Inst.*, vol. vi., page 421.

observe with surprise and regret that the Institute has seen fit to publish this paper without any reference to this company, seeing that it deals not only with scientific matters of interest to an Institute such as yours, but enters into much wider questions with which your Institute can have no concern. Amongst these are the constitution and financial position of the company, its prospecting operations, and their results outside of the Singareni coal-field. Even in dealing with the coal-field, of which alone Mr. Kirkup could have any personal knowledge, he has thought fit to set forth particulars as to the cost of raising coal, the prices obtained from different customers and contractors, the trading prospects, etc., all of which information must have been obtained by him in his confidential capacity of an employé of the company, who was bound by the terms of his agreement not to divulge any matters relating to the company's business. Apart from the publication of figures which Mr. Kirkup should not have made use of without the consent of the Board, and some statements which are inaccurate, relating to the colliery business, he has imported into the paper various references to operations in other districts, having no connexion with the subject of his paper. The Board will not here discuss every misstatement contained in Mr. Kirkup's paper, but will take as an instance the very first page of that paper, in which the following remarks appear :—

> A considerable amount of capital has been expended in prospecting work at the so-called Golconda diamond-mines (situated on the banks of the Kristna river) which have proved unsuccessful, and also in small areas of gold-bearing schists in the south-western part of the territory. Some old workings have been partially explored, but up to the present time no very rich quartz has been found. The remainder of the territory has been practically unprospected by the company.*

As a matter of fact, a large number of diamonds have been found. The areas of gold-bearing schists in the south-western portion of the territory prospected are not small but very extensive. An enormous number of old gold-workings have been discovered, and many of them deeply explored by the company. One of the old shafts bottomed extends to 385 feet below the surface, being the deepest ancient gold-mine shaft yet found in India. Very rich quartz has been discovered in various places. A very large area of territory, outside that containing gold-bearing schists, has been carefully prospected, and hundreds of square miles have been selected, and leases applied for, which the Native Government, under its concession to the company, is bound to grant. There is no justification for Mr. Kirkup's † remarks that the expenditure

* *Trans. Fed. Inst.*, vol. vi., page 421. † *Ibid.*, page 442.

has been exceptionally high under "unqualified and amateur superintendence in the early days of the colliery." As to this expression, it may be remarked that the whole of the mines established by this company from the beginning of their operations have answered their purpose and have proved profitable except one, viz., No. 6 incline. That one was opened under Mr. Kirkup's superintendence, it proved a failure, and was abandoned. Mr. Kirkup's remarks that "the present mines will not satisfy the requirements for 1893,"* and that "it is doubtful, even if they should, whether the contractor will be able to dispose of the largely increased quantity,"† are inconsistent with each other. It is the fact that the company were enabled to satisfy the demand during 1893, and have since raised the output to a considerably higher point than was ever previously attained, the present average output exceeding 4,000 tons per week. There are many other passages in Mr. Kirkup's paper, especially those that refer to costs and selling prices of coal, and its steam-producing value, which are incorrect, and there are other remarks that are irrelevant and uncalled for.

The CHAIRMAN said that before the meeting dispersed they would agree with him in proposing a hearty vote of thanks to the owners of the collieries and mines which had been so kindly thrown open to the inspection of the members. He was sure that they had all enjoyed coming to this district. It was a long time since they had held a meeting here, but they had been received so kindly and courteously that he knew he was only expressing the feelings of all in tendering their best thanks for the facilities extended to them.

The resolution was carried with acclamation, and the meeting terminated.

The following papers record some of the features of interest seen by visitors to collieries and mines which were, by kind permission of the owners, open for inspection during the course of the Whitehaven meeting, on June 21st and 22nd, 1894 :—

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* *Trans. Fed. Inst.*, vol. vi., page 443. † *Ibid.*, page 443.

DESCRIPTION OF THE WHITEHAVEN COLLIERIES.

By H. M. JAMES.

The Whitehaven collieries are the property of the Earl of Lonsdale, and are being worked under lease by the Whitehaven Colliery Company. The William, Wellington, and Croft pits are each sunk 840 feet to the six-quarters coal-seam, and each work the main band coal-seam at a depth of 600 feet. With the exception of a small district in the Croft pit, leased from the trustees of St. Bees grammar school, the whole of the coal is worked under the sea, the average distance from the pit's bottom to the forehead being 3 miles.

The coal is brought from the faces by different systems of haulage in the different pits.

At the William pit, the coal is brought from the Countess and Delaval districts by compressed air-engines placed at the top of the brows leading to those districts, and bringing the coal up by the endless-rope system to the main engine-plane, whence by combinations of the endless and tail-rope systems the coal is brought in sets of from seventy to eighty tubs to the pit-bottom. It is intended to make this main engine-plane, which is now a single road, into a double one very shortly, the necessary work being in progress. The output of this pit is 800 tons per day of ten hours.

The Wellington pit is situated on the south side of Whitehaven harbour. The main engine-plane, which dips at an angle averaging 1 in 10, is laid with a double road. At a distance of 9,000 feet in-bye the main haulage-rope works friction-gear, with two drums, from which drums the haulage-rope is taken into the two districts into which the workings are divided. The tubs, brought out by means of the friction-gear, are made up in sets and taken to the pit-bottom along the main engine-plane by the endless-rope system of haulage in sets with bogies at either end. This pit also has an output of about 800 tons per day.

The Croft pit is situated 1½ miles to the south of the Wellington pit, with which it is connected by a tramway along the cliffs. The workings are divided into several districts, whose roads converge on to the main

engine-plane at the same point. From one district the coal is hauled, by friction-gear, on the endless-rope system with underneath attachment, and in another by the endless-rope system with overhead attachment. The friction-gear is driven by the main haulage-rope, and the tubs, when delivered on to the main engine-plane, are made up in sets and brought to the pit-bottom on the endless-rope system in sets with one bogie in front. The output of this pit is 600 tons per day of ten hours.

In each case the underground hauling is effected by engines placed at the top of the pits. An interesting feature at the William pit is the Newcomen and Watts atmospheric engine doing efficient work, the vacuum being produced from the exhaust steam of the fan-engine.

The pits are all ventilated by Guibal fans.

There are 76 beehive coke-ovens at work at the William pit, and 50 additional coke-ovens are being built. The small coal is washed, and ground in a Carr disintegrator before being put into the coke-ovens. The produce is a hard coke, well suited for blast-furnace purposes. Six boilers are attached to the existing coke-ovens, and three more are being fixed to the coke-ovens in course of erection, the waste heat from the coke-ovens being thus utilized to material advantage.

DESCRIPTION OF THE ST. HELEN'S COLLIERY, WORKINGTON.

By GEORGE SCOULAR.

The proprietors of the colliery are the St. Helen's Colliery and Brick Works Co., Ltd., of Workington. The lessor is the Earl of Lonsdale. There are two shafts 10 feet 9 inches and 11 feet 8 inches in diameter respectively, sunk close together on the sea-coast at Siddick station, about 1 mile north-east from Workington harbour. The royalty, not including that under high-water mark, extends to about 2,600 acres.

The seams being worked are the ten-quarters, cannell and metal, little main, and lickbank; the depths being, respectively, 504, 714, 918, and 1,020 feet. The present area of worked ground is about 200 acres. The ten-quarters seam yields about 2 feet 8 inches of clean coal, the cannell and metal about 7 feet, the little main 2 feet 2 inches, and the lickbank about 2 feet 2 inches. The seams, especially the cannell and metal, are intermixed with bands of dirt, and are also, as is general in the Cumberland coal-field, considerably faulted.

The seams are worked on the longwall system, with the exception of the cannell and metal, which is worked on the pillar-and-stall system.

Coal is raised at both upcast and downcast shafts. The winding-engine at the downcast shaft has two cylinders 30 inches in diameter and 5 feet stroke, with Cornish valves and a cylindrical drum 15½ feet in diameter. The ropes are 1½ inches in diameter. The cages are three-decked, one tub carrying 11½ cwts. being on each deck. At the upcast shaft the winding-engine has two cylinders 26 inches in diameter and 5 feet stroke, with drop valves and a cylindrical drum 14 feet in diameter. The cages are double-decked, with one tub on each deck.

The pumping-engine, which is of the Cornish type with overhead beam, has a cylinder 60 inches in diameter and 9 feet stroke, and actuates two lifts of pumps each 252 feet long, the upper one being 18 inches in diameter and the lower one 16 inches in diameter.

The Guibal fan is 36 feet in diameter by 12 feet wide, and is driven by a single non-condensing engine, with duplicate cylinders.

The hauling-engines are horizontal compound-condensing, with cylinders 16 inches and 30 inches in diameter by $2\frac{1}{2}$ feet stroke, and are erected on the surface. They transmit their power by two band-ropes to the haulage stations underground in the lickbank and the cannell and metal seams. In the lickbank seam one of the band-ropes drives four clutch-pulleys, which are mounted on a vertical shaft, and gives motion to endless-ropes running at the rate of $2\frac{1}{2}$ miles per hour along the roadways radiating from the haulage-station. In the cannell and metal seam there is a similar arrangement of three clutch-pulleys driven by the other band-rope.

There are two air-compressing engines, each having one steam and one air-cylinder 24 inches in diameter by 4 feet stroke : the compressed-air being used underground for dip-haulage and pumping.

For screening and picking the coal, there are three jigger-screens, each discharging the round coal upon a separate travelling table, where the dirt is hand-picked. The small-coal from each screen is conveyed by a travelling-belt to the small-coal bunker, from whence it is elevated to a revolving riddle, where it is sized, and is afterwards washed in the Coppée washery ; the nut and coking coal being automatically discharged into the several bunkers, ready for loading into railway-waggons or into the coke-oven bogies.

A battery of twenty-four Coppée—forty-eight hours, 6 tons—coke-ovens has recently been put into operation, the gases from which fire two Babcock and Wilcox water-tube boilers.

HISTORY AND DESCRIPTION OF THE GREENSIDE SILVER LEAD MINE, PATTERDALE.

By W. H. BORLASE.

Greenside is one of the oldest, if not the oldest, silver lead mine now at work in the United Kingdom. Its history up to the "eighties" must have been one of gratifying success, but of late the prices of silver and lead have materially militated against results.

The original workings are situate on a tableland and slope of one of the spurs of Helvellyn, starting at an elevation of 1,850 feet above sea-level, and through a hill rising to the north, having an altitude of 2,500 feet. The washing and dressing appliances were of the most primitive type, and were only workable nine months in the year, frost and snow interfering during the winter months with this operation. The ore at that time was conveyed to the neighbourhood of Keswick on the backs of horses for smelting, but for the last sixty years the company has itself smelted the produce, desilverizing the same, and selling it in soft lead pigs and pure silver cakes.

The predominating geological formation is porphyritic, being largely made up of syenite, felspathic ashes, and traps, which more or less intrude on and into the vein in different sections of the mine.

The composition of the vein varies considerably, in sympathy with its structural surroundings. The ore-bearing sections are embedded in a hard massive (either orthoclastic or plagioclastic) felspar, in which it is composed of quartz, carbonate of lime, sulphate of baryta, with blende (sulphide of zinc) thinly dispersed, and argentiferous lead-ore. When baryta is present the vein generally opens to a great width, yielding in some cases 10 tons of silver lead-ore per fathom. The non-productive sections are encased in a less compact and softer series, the vein itself being reduced to small proportions, and principally composed of flookan and friable quartz, with small quantities of iron pyrites. The bearing of the vein is about 20 degs. east of north, with an underlie or inclination dip to the east of 18 degs. from the vertical line.

The southern portion of the workings is singularly disordered by a cross-course, locally termed "crossback" or clay vein, which though small in size when compared with the Greenside vein (which it crosses nearly at right angles), has, nevertheless, so mastered the latter that it is broken up and split into branches, and has never been recognized south of the inter-

section. The north section of the mine has also a crossback or cross-course, but the result at the crossing is so completely reversed that one is struck with astonishment to see the opposites in the same formation. But on studying the conditions of the intersections the difference may, to some extent, be accounted for. The former bears south-east by north-west, with a southern underlie, carrying a good hanging wall or cheek, and composed of soft clay flookan. The latter (north cross-course) is small, well defined, bearing south-west by north-east, with a slight dip northwards, bringing with it from the west a hard and well-defined greenstone.

To the north of this intersection—the effect of which is that the vein is carried away nearly due east for 60 feet, after which it gradually recovers its proper bearing—the vein opening out to a width of 16 feet, is composed of friable quartz, carbonate of lime, and is very productive of lead-ore. Here we meet with a bastard granite or syenite, very hard at times, and at others partially decomposed, resembling very much the china-stone of Cornwall, the vein being more productive in the harder portions, the softer portions giving a great deal of labour and trouble when the stuff is being removed from the headings and stopes, the hanging side exfoliating extensively. This action occurs more especially where the vein throws off what are known as the east branches, or may we say where the feeders of the vein come in from the east; for in close proximity to these junctions there is invariably a good deposit of ore.

The vein at times carries large quantities of felspar, having a flint-like appearance, quite phonolitic, and may be termed clinkstone.

Up to the year 1869, the workings were carried on through the various day-levels, the highest starting at 2,000 feet, the lowest, the Lucy Tongue level, at 1,100 feet above sea-level, and they are mostly cross-cuts. The last-named was started on a vein, known as the Lucy Tongue vein, which runs parallel to the Greenside vein, and the driving was continued on its course until the south cross-course or clay vein, already mentioned, was intersected, and being a loop for the two veins was made use of, the level vein turned and driven on it for a considerable distance; but the bearing being too southerly for the ore-producing zone proved in the upper levels, the direction was again changed to north-west, to come under the productive ground; consequently the last portion of this level is in country rock. Seventeen years were occupied in this driving, the whole distance being $\frac{3}{4}$ mile, and considering that there were no means of ventilation beyond a small fan-blast, one is led to wonder how the work was accomplished. We are certain that such proceedings would not be allowed at the present day.

From the level driven at the altitude of 1,600 feet, a shaft is sunk to the base of the Lucy Tongue level 480 feet below, and the ground intervening is considerably wrought, the haulage being done by a hydraulic double-cylinder engine, receiving its power from a store of water 180 feet above, giving a working pressure of 80 lbs. per square inch. This, for many years, was the only engine employed in the mine, and until the Lucy Tongue level was driven up to the vein, and a shaft sunk on the point of intersection, no other was needed. The No. 2 hydraulic engine was erected to work this shaft, receiving its power from the exhaust water of the No. 1 engine, having an effective head of 480 feet or 212 lbs. pressure to the square inch.

There are three shoots, so-called, or deposits of ore in the developed property. The south and middle deposits have been somewhat erratic in their productiveness, either very good or very poor. But the north shoot, which was first discovered in the high mountain-lands, has been continuous, apparently improving as depth is attained, and judging from the present appearances of the vein and the rocks in which it is encased, the author ventures without hesitation to assert that great riches still remain for deeper sinking.

The middle bunch (or deposit) showed at first in the low level, but became poor about 60 feet below and remained so until 78 feet below the Lucy Tongue level was reached, where it made a splendid deposit of ore for 150 feet in length, suddenly cutting out as suddenly as it came in. Compactness in the surrounding rock in this section is singularly absent when the vein is non-productive, and probably accounts in some measure for the erratic character of the vein here, for as soon as the ground assumes its normal hardness productiveness is resumed.

The same remarks will more or less apply to the southern deposit or bunch of ore, excepting that the intermittent productiveness must be credited to the near proximity and influence of the cross-course. Throughout the mine the ore-bearing ground has materially lengthened from the Lucy Tongue level downwards, the vein being more compact and the ore much more concentrated.

The levels are usually driven 7 feet high by 7 feet wide, over which another level is driven, leaving an arch or roof of 5 or 6 feet between them. This plan is adopted to save wooding. Through the arches or roofs hopper-holes are sunk about 18 feet apart, and shoots put in to fill the stuff direct from the headings into the waggons, from whence the same waggons deliver their contents to the washing and picking-grates. The headings are continued from roof to sole of the levels, and are worked in sections or lifts of 12 feet high each.

The 120 feet level below the Lucy Tongue level is only driven through the southern deposit of ore, but the 240 feet level is driven as far north as the Lucy Tongue level, and still continues in good ore. At this level, near the No. 2 shaft is fixed a Cameron pump driven by compressed air, supplied from outside the mine, a distance of nearly a mile, and it was found quite adequate to deal with the whole of the water in the mine at this depth; but when sinking for, and the driving of a 360 feet level was required, this power was no longer available, unless at the expense of the rock-drills and sump winding pneumatic engines. As we were fully convinced that an extra or supplemental plant must be provided, the question arose of compressed air *versus* electricity.

With the former we had already some experience and were aware of its capabilities, and after mature deliberations in matters of site for the initial power to be employed to generate the extra power needed inside the mine, and to avoid robbing any of the existing machinery, the author decided in favour of electricity. He accordingly recommended the new power to his directors, who readily entertained the suggestions, saw the advantages offered, and in due time ordered the work to proceed.

The following description of the plant conveys some idea of its extent at work and provided for, its power capabilities and utility.

On the eastern slopes of Helvellyn lies a small natural lake called Red Tarn, and on the north-east another called Keppelcove Tarn. Between these two waters rises the hill of Catchedecam, at the base of which the overflows join, and near to which the turbine-dynamo station is erected. The water is taken at an elevation of 1,750 feet above sea-level, and flows through an open watercourse $1\frac{1}{4}$ miles in length to a wooden tank, from which it is conveyed down the hillside for a distance of 1,080 feet, in cast-iron pipes 15 inches in diameter. The fall at the station is equivalent to a vertical head of 400 feet, and the effective horse-power is about 200.

The generating-station contains a Gilkes vortex turbine of 100 horse-power, driving a four-pole compound-wound dynamo, and provision is made to duplicate this plant when necessary; pipes, watercourses, etc., are already laid for this purpose accordingly.

The electric current is conveyed by two bare copper conductors on poles for $\frac{3}{4}$ mile, where it enters the mine at an elevation of 1,600 feet above sea-level. The conductors from this point are insulated and covered with lead. About $\frac{3}{4}$ mile in the mine and down through No. 1 shaft, or $1\frac{1}{2}$ miles from the dynamo, a 9 horse-power series-wound motor is winding the stuff from a pare of sinkers. Farther into the mine, another $\frac{1}{4}$ mile and down through No. 2 shaft to the 360 feet level or bottom of

the mine, is fixed another 9 horse-power motor working a three-throw pump, forcing the whole of the water in the mine below the Lucy Tongue level, up to that level, 360 feet high.

About midway between No. 1 and No. 2 shafts is fixed a transformer, which reduces the pressure from 600 volts to 250 volts, for the purpose of working an electro-locomotive in the lowest day-level, through which runs the water used by the two hydraulic engines. The locomotive runs with a train of twelve waggons, the total weight when loaded being 18 tons (exclusive of itself), and it does the work with the greatest ease over gradients in places of as much as 1 in 20 and 24. The conductors in this day-level are of phosphor-bronze wire, and the current is fed to the locomotive by four contact-pulleys. The difficulties encountered in fixing this plant and wiring the level can only be appreciated by the practical miner. The curves are so numerous that the train occupies three and four of them at a time in places. The gauge of the railway is 22 inches, width of locomotive 32 inches over all, and height 46 inches.

The engine-rooms, caging-stations at the different levels, and shaft tops, are lighted by six 100 volts 16 candle-power lamps, wired in series. The adoption of this power has enabled the mine to be worked with six horses less underground, and has effected considerable saving in other ways. The cost of maintenance since December, 1891, including accidental breakages and wearing renewals, *i.e.*, brushes, etc., has not exceeded £25, which speaks for itself, when taking into consideration that no professional electrician is employed on the staff.

The turbine-dynamo station, and offices outside, are in communication with the chief stations inside the mine by telephone.

The dressing department is equipped with Blake stone-breakers, from which the stuff gravitates to Cornish and other crushers, from them through revolving-trommels and V-classifiers, which supply Green and other continuous jiggers, the finer stuff and slimes being taken to larger V-classifiers, and from them to automatic and other round buddles. The ore is dressed up to 80 and 82 per cent. of lead.

The smelt-mill is provided with roasting and other calcining-furnaces, Scotch hearths being used for smelting, and the Pattinson process for desilverizing.

Few mines are so advantageously situated for cheap power as this one, consequently many are being compelled to cease working owing to the very low prices which have now continued so long. But the directors of this company have still a hope that the once flourishing and important industry of lead-mining in this country will not become extinct.

A SHORT DESCRIPTION OF THE HÆMATITE DEPOSIT WORKED BY THE SALTER, ESKETT, AND WINDER GILL MINES, AND THE METHOD OF WORKING IT.

By J. D. KENDALL.

The ore worked by these mines forms one large deposit lying alongside a fault bearing about 34 degs. north-west and south-east, from the true meridian. The ore occurs mainly in the fourth limestone, but also partly in the third, sixth, and seventh limestones. In form it is mainly vein-like, but has one or two bed-like extensions from it into both the hanging and lying walls. The length of the deposit is about 2,880 feet, its greatest width 180 feet, and its greatest depth on the hade of the fault about 315 feet. The quantity of ore yielded by the deposit is between 4 and 5 million tons. Like many other similar deposits, it is best developed near the outcrop, and becomes gradually reduced in size and importance as it is followed to the dip.

The throw of the fault alongside which the ore occurs is about 660 feet up to the west, the rocks on opposite sides of the dislocation having been shifted thereby about 4,000 feet. About 400 feet of the vertical movement has occurred in post-Permian times. The dip of the enclosing rocks is to the north-west, and the main dip of the ore appears to be along the line of the fault at an angle of about 1 in 6 from the horizontal: the depth of the bottom of the deposit at the north end being about 700 feet, whilst at the south end it is only about 200 feet. A large part of the ore, however, has also a dip to the east, due to the hade of the fault, so that the real dip of the vein-like part of the deposit is nearly due north.

In working the deposit, 13 shafts have been sunk at different parts of its length, the deepest being 650 feet. From these shafts, levels are driven out at suitable depths, generally about 60 feet apart vertically, and the deposit is worked, as is usual with such deposits, on the pillar-and-stall system. The ore is very hard and loughy, and therefore more expensive to work than ores of a softer nature, such as occur in the southern part of the district, although it does not require so much timber in the first working.

The water that finds its way into the mines, and which has to be pumped, varies with the seasons from 120 to 450 gallons per minute.

APPENDICES.

I.—BAROMETER, THERMOMETER, Etc., READINGS FOR THE YEAR 1893.

By M. WALTON BROWN.

The barometer, thermometer, etc., readings have been supplied by permission of the authorities of the Glasgow and Kew Observatories, and give some idea of the variations of temperature and of atmospheric pressure in the intervening districts in which the mining operations of this country are chiefly carried on.

The barometer at Kew is 34 feet, and at Glasgow is 180 feet, above sea-level. The barometer readings at Glasgow have been reduced to 32 feet above sea-level, by the addition of 0·150 inch to each reading, and the barometer readings at both observatories are reduced to 32 degs. Fahr.

The fatal explosions in collieries are obtained from the annual reports of H.M. Inspectors of Mines, and are also printed upon the diagrams (Plates XXII. and XXIII.) recording the meteorological observations.

List of Fatal Explosions in Collieries during 1893.

1893.	Colliery.	District.	Deaths.
Feb. 11, 11·15 a.m. ...	Whitwick (No. 6) ...	Midland	2
„ 11, „ ...	Werfa	South Wales	1
April 21, 7·20 „ ...	Himley	Staffordshire, South ...	2
May 1, 6·20 „ ...	Sharlston	Yorkshire	3
„ 5, 10·30 „ ...	Fishley	Staffordshire, South ...	1
„ 24, 10· 0 p.m. ...	Dunston	Newcastle	2
June 8, 2·30 „ ...	Herbertshire (No. 3) ...	Scotland, West ...	2
„ 30, 9· 0 a.m. ...	Butterknowle	Durham	1
July 4, 12· 3 p.m. ...	Combs	Yorkshire	139
Aug. 16, 8· 0 „ ...	Hattonrig (No. 3) ...	Scotland, West ...	1
Oct. 24, 7· 0 a.m. ...	Elms	Liverpool	1
Nov. 13, 6· 0 „ ...	Cowdenbeath	Scotland, East ...	1
„ 13, p.m. ...	Camerton	South-Western ...	2
Dec. 2, 12·45 „ ...	Hem Heath	Staffordshire, North ...	1
„ 8, 10·15 a.m. ...	Wooley	Durham	1
		Total	160

Deaths from Explosions of Fire-damp during 1893, in the several Inspection Districts.

	Deaths.
Durham	2
Liverpool	1
Manchester	0
Midland	2
Newcastle	2
Scotland, East	1
Do. West	3
South Wales	1
South-Western	2
Staffordshire, North	1
Do. South	3
Yorkshire	142
Total	160

JANUARY, 1893.

KEW.								GLASGOW.							
	BAROMETER.				TEMPERATURE.				BAROMETER.				TEMPERATURE.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	29·859	29·875	29·850	29·896	30·2	21·8	NE	1	29·984	30·028	30·068	30·099	35·4	28·4	ESE
2	29·922	29·973	30·036	30·080	28·0	20·6	N	2	30·116	30·130	30·084	29·997	31·5	24·9	NW
3	30·097	30·142	30·166	30·245	30·9	20·7	W	3	29·991	30·112	30·194	30·264	34·4	28·7	NNE
4	30·301	30·378	30·394	30·397	23·9	16·2	W	4	30·294	30·328	30·347	30·342	31·9	29·6	E
5	30·392	30·397	30·336	30·297	26·1	13·1	W	5	30·329	30·338	30·303	30·260	31·4	16·6	E
6	30·228	30 202	30·091	30·005	31·6	26·0	E	6	30·176	30 123	30·027	30·019	31·8	16·9	E
7	29·907	29·858	29·795	29·763	34·2	27·8	E	7	30·002	29·992	29·953	29·953	34·9	29·3	E
8	29·736	29·687	29·652	29·675	35·4	33·2	ENE	8	29·925	29·863	29·794	29·811	37·1	33·6	
9	29·673	29·709	29·727	29·808	38·2	34·9	NE	9	29·815	29·874	29·922	30·006	37·1	34·6	ENE
10	29·872	29·983	30·050	30·172	39·8	33·7	NNE	10	30·073	30·182	30·268	30·358	38·1	34·6	ENE
11	30·212	30·295	30·261	30·199	37·2	31·7	NNE	11	30·395	30·431	30·359	30·347	39·0	29·8	W
12	30·137	30·180	30·174	30·173	36·7	34·5	N	12	30·284	30·260	30·190	30·164	37·9	26·4	W
13	30·103	30·054	29·961	29·913	40·9	34·3	WNW	13	30·086	29·988	29·936	29·891	44·0	32·4	W
14	29·789	29·703	29·611	29·786	37·6	32·9	WNW	14	29·769	29·720	29·915	30·083	36·4	29·6	NNE
15	29·992	30·157	30·222	30·219	33·5	25·3	NNE	15	30·164	30·220	30·114	29·944	36·1	28·0	WSW
16	30·073	29·889	29·597	29·599	40·6	24·9	SSW	16	29·739	29·670	29·720	29·834	47·0	35·7	WNW
17	29·713	29·845	29·915	30·069	39·1	30·2	SSE	17	29·784	29·813	29·918	29·916	41·8	35·3	W
18	30·099	30·121	30·211	30·337	44·3	29·0	SE	18	29·869	30·002	30·143	30·209	48·3	40·2	SW
19	30·375	30·410	30·373	30·355	41·6	36·7	WNW	19	30·158	30·118	30·035	30·126	48·2	40·8	SSW
20	30·308	30·340	30·347	30·400	43·0	36·5	WNW	20	30·135	30·106	30·179	30·271	41·3	35·4	WNW
21	30·404	30·376	30·356	30·378	43·0	33·9	NW	21	30·238	30·297	30·310	30·279	45·1	35·1	W
22	30·321	30·295	30·214	30·145	44·6	38·5	WNW	22	30·309	30·163	30·084	30·020	49·4	42·3	WSW
23	30·071	30·068	30·082	30·118	50·5	42·1	W	23	29·943	29·947	29·923	29·902	49·7	46·5	WSW
24	30·117	30·133	30·097	30·097	47·3	43·9	WSW	24	29·830	29·738	29·689	29·779	48·9	43·7	SW
25	30·125	30·179	30·140	30·085	45·4	37·7	WSW	25	29·862	29·931	29·928	29·808	45·0	39·1	SW
26	29·964	29·894	29·778	29·763	45·8	39·0	S	26	29·482	29·378	29·573	29·637	45·7	38·7	WSW
27	29·902	30·030	30·009	29·974	40·3	27·4	N	27	29·697	29·809	29·808	29·755	45·8	38·7	SW
28	29·844	29·793	29·707	29·631	44·3	34·9	S	28	29·594	29·472	29·355	29·317	43·8	34·8	E
29	29·484	29·443	29·527	29·780	50·9	40·8	S	29	29·176	29·226	29·357	29·538	47·1	40·9	SSW
30	29·930	30·015	30·006	29·967	50·3	39·9	S	30	29·616	29·662	29·594	29·512	49·9	41·9	S
31	29·899	29·930	29·937	29·889	52·2	46·9	SW	31	29·480	29·451	29·576	29·624	48·8	41·6	SW

FEBRUARY, 1893.

1	29·705	29·732	29·972	30·069	47·9	41·8	W	1	29·595	29·717	29·765	29·687	44·5	36·0	W
2	29·965	29·857	29·952	30·081	53·5	41·8	W	2	29·595	29·641	29·793	29·917	44·1	36·6	W
3	30·144	30·203	30·239	30·284	48·1	43·3	SW	3	29·950	30·027	30·045	30·089	49·3	39·6	SSW
4	30·309	30·349	30·347	30·380	44·5	33·5	SE	4	30·098	30·133	30·176	30·220	47·0	43·6	SSW
5	30·375	30·396	30·352	30·366	39·0	27·6	SE	5	30·213	30·200	30·171	30·169	44·0	39·8	S
6	*	30·372	30·319	30·288	41·3	23·0	SSE	6	30·130	30·089	29·981	29·799	46·3	39·2	S
7	30·185	30·211	30·203	30·157	52·2	37·9	W	7	29·819	29·873	29·877	29·635	47·3	41·7	WSW
8	29·927	29·888	29·851	29·929	47·5	35·9	WSW	8	29·388	29·600	29·641	29·651	43·4	37·0	W
9	29·931	29·895	29·715	29·411	48·7	34·8	SW	9	29·679	29·421	29·081	28·930	45·2	38·0	SW
10	29·360	29·306	29·531	29·754	51·3	41·9	W	10	28·774	28·996	29·372	29·331	48·1	38·1	WNW
11	29·700	29·626	29·611	29·643	51·2	38·1	W	11	29·203	29 237	29·302	29·317	45·4	36·7	WSW
12	29·637	29·691	29·789	29·869	44·9	34·5	W	12	29·325	29·577	29·627	29·648	38·8	32·9	W
13	29·810	29·728	29·605	29·515	48·5	35·2	SW	13	29·584	29·483	29·272	29·175	36·9	31·2	ESE
14	29·397	29·319	29·288	29·527	52·1	39·5	SSW	14	29·131	28·749	28·792	29·054	46·8	34·0	S
15	29·664	29·732	29·679	29·618	48·8	34·7	SE	15	29·277	29·492	29·519	29·497	44·4	36·2	SSW
16	29·561	29·544	29·617	29·710	50·3	40·4	S	16	29·391	29·344	29·330	29·446	46·9	37·7	S
17	29·819	29·982	30·031	30·038	45·9	36·0	SW	17	29·553	29·686	29·725	29·668	41·7	35·3	SW
18	30·020	30·052	29·986	29·961	53·5	43·2	SW	18	29·504	29·594	29·633	29·657	49·4	37·1	SW
19	29·863	29·820	29·681	29·610	56·3	47·3	S	19	29·618	29·583	29·483	29·420	51·7	46·1	SE
20	29·601	29·526	29·482	29·347	48·2	40·2	SW	20	29·360	29·363	29·272	29·150	51·1	39·3	SW
21	28·895	28·710	28·671	28·785	48·8	40·3	S	21	29·007	28·965	28·976	29·055	43·3	36·2	NE
22	28·885	28·981	29·083	29·242	40·7	35·2	WNW	22	29·164	29·328	29·416	29·500	41·9	34·3	NE
23	29·342	29·434	29·462	29·459	35·3	32·8	NE	23	29·529	29·558	29·515	29·463	39·3	32·7	ENE
24	29·389	29·308	29·216	29·181	37·9	31·0	E	24	29·371	29·308	29·218	29·219	38·3	30·5	NW
25	29·155	29·236	29·327	29·385	41·7	29·9	NNW	25	29·200	29·233	29·243	29·280	36·4	26·7	N
26	29·207	28·923	28·922	29·045	47·2	36·9	ESE	26	29·188	29·036	28·747	28·729	34·1	30·2	NE
27	29·229	29·327	29·263	29·341	41·2	35·6	SE	27	28·890	29·061	29·196	29·374	40·3	31·8	W
28	29·524	29·790	29·938	30·019	44·8	34·2	N	28	29·502	29·589	29·670	29·739	39·1	26·3	SW

* Gas out.

MARCH, 1893.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	29·856	29·580	29·566	29·730	55·7	32·7	SW	1	29·597	29·289	29·213	29·220	37·9	31·0	NE
2	29·794	29·921	29·989	30·113	56·0	46·3	W	2	29·467	29·744	29·941	30·105	44·1	35·2	NW
3	30·202	30·291	30·258	30·210	48·2	42·4	SE	3	30·125	30·081	29·924	29·830	45·9	34·9	NE
4	30·144	30·138	30·117	30·186	56·1	42·0	SW	4	29·779	29·828	29·864	29·900	48·9	44·6	SW
5	30·165	30·176	30·196	30·261	55·8	41·0	WNW	5	29·915	30·011	30·063	30·167	52·1	45·6	W
6	30·266	30·285	30·220	30·261	51·4	41·7	WNW	6	30·137	30·089	30·123	30·169	51·1	43·7	SW
7	30·301	30·321	30·276	30·290	53·4	39·2	W	7	30·093	30·094	30·123	30·204	52·9	44·7	SW
8	30·286	30·332	30·271	30·265	61·0	41·8	WNW	8	30·210	30·230	30·169	30·130	50·1	42·1	WSW
9	30·211	30·182	30·094	30·095	57·4	38·4	SW	9	30·018	29·899	29·868	29·947	50·7	39·0	WSW
10	30·195	30·285	30·284	30·350	51·7	38·8	NW	10	29·992	30·106	30·181	30·210	47·8	37·6	NW
11	30·351	30·321	30·203	30·157	54·2	31·9	SW	11	30·079	29·958	29·840	29·794	45·7	40·8	S
12	30·041	29·969	29·786	29·703	59·4	30·7	S	12	29·718	29·629	29·493	29·414	47·6	42·9	SW
13	29·632	29·702	29·700	29·733	55·1	35·3	NW	13	29·511	29·608	29·635	29·652	48·1	37·0	W
14	29·723	29·761	29·788	29·800	52·4	45·8	W	14	29·538	29·494	29·417	29·398	48·5	38·1	SW
15	29·796	29·805	29·756	29·748	53·9	46·7	SW	15	29·358	29·361	29·369	29·318	48·2	36·8	SW
16	29·668	29·634	29·659	29·773	52·5	37·9	WSW	16	29·271	29·354	29·415	29·476	41·3	30·1	WNW
17	29·806	29·834	29·919	30·065	45·4	35·3	W	17	29·456	29·696	29·835	29·944	38·5	29·8	W
18	30·070	30·150	30·238	30·367	42·5	29·9	N	18	30·046	30·188	30·260	30·330	39·2	29·2	NW
19	30·407	30·446	30·398	30·412	45·8	26·9	N	19	30·331	30·340	30·306	30·301	40·0	28·0	NW
20	30·407	30·437	30·359	30·371	52·9	28·6	SSW	20	30·284	30·305	30·277	30·275	49·0	39·6	SW
21	30·371	30·376	30·314	30·333	54·8	28·1	SW	21	30·253	30·264	30·213	30·241	54·3	33·1	W
22	30·302	30·297	30·225	30·241	53·9	34·6	E	22	30·214	30·201	30·157	30·158	56·9	34·4	SW
23	30·233	30·269	30·234	30·283	61·9	38·0	ENE	23	30·150	30·168	30·157	30·204	57·9	34·6	SSW
24	30·304	30·350	30·334	30·394	61·0	32·6	E	24	30·228	30·277	30·266	30·325	60·9	34·6	SW
25	30·404	30·422	30·353	30·354	59·0	30·4	E	25	30·342	30·373	30·336	30·326	60·5	35·3	SW
26	30·303	30·261	30·157	30·134	54·0	28·9	E	26	30·288	30·264	30·209	30·235	56·5	38·7	E
27	30·087	30·083	30·046	30·101	56·6	34·0	ENE	27	30·215	30·195	30·168	30·200	46·9	39·6	E
28	30·111	30·126	30·091	30·110	56·6	38·9	E	28	30·187	30·193	30·112	30·124	50·0	35·4	ENE
29	30·070	30·064	29·992	30·030	64·9	35·8	E	29	30·070	30·040	29·920	29·890	60·0	32·2	E
30	29·990	29·965	29·853	29·864	64·2	32·6	ESE	30	29·825	29·827	29·736	29·675	54·0	42·9	W
31	29·838	29·851	29·786	29·841	63·1	34·7	SW	31	29·587	29·563	29·537	29·645	51·2	43·2	SSW

APRIL, 1893.

1	29·882	29·927	29·910	29·972	67·2	33·9	SE	1	29·755	29·893	29·935	30·022	50·8	41·1	W
2	30·051	30·152	30·163	30·249	60·0	42·8	N	2	30·057	30·129	30·126	30·145	53·2	37·6	SW
3	30·236	30·232	30·176	30·210	61·1	38·6	ENE	3	30·148	30·176	30·167	30·211	54·6	42·8	WSW
4	30·203	30·227	30·187	30·199	59·1	36·7	E	4	30·226	30·247	30·183	30·197	57·4	33·3	NE
5	30·167	30·161	30·130	30·152	55·1	44·7	NE	5	30·167	30·141	30·052	30·082	64·8	36·1	S
6	30·137	30·186	30·175	30·261	56·1	43·3	ENE	6	30·106	30·171	30·131	30·149	63·2	41·7	NE
7	30·306	30·354	30·332	30·409	64·4	38·9	NE	7	30·347	30·423	30·411	30·500	55·4	39·3	E
8	30·445	30·484	30·399	30·422	57·4	39·8	NE	8	30·519	30·568	30·521	30·542	54·6	42·0	E
9	30·411	30·406	30·290	30·292	57·0	39·1	NE	9	30·469	30·427	30·297	30·311	59·5	35·6	N
10	30·244	30·211	30·116	30·144	64·6	37·5	NE	10	30·292	30·307	30·263	30·305	52·3	34·1	E
11	30·181	30·254	30·263	30·346	48·0	38·8	NE	11	30·331	30·406	30·402	30·418	47·0	40·7	E
12	30·341	30·365	30·299	30·293	47·8	34·3	NNE	12	30·395	30·372	30·294	30·309	54·6	34·1	NW
13	30·265	30·309	30·314	30·372	50·2	32·2	N	13	30·319	30·355	30·294	30·296	53·2	42·1	NNE
14	30·368	30·369	30·257	30·257	56·1	30·8	W	14	30·232	30·183	30·120	30·088	52·8	39·6	WNW
15	30·214	30·215	30·115	30·148	61·8	32·6	SW	15	29·987	29·962	29·953	29·973	51·1	45·0	W
16	30·135	30·145	30·096	30·118	60·3	38·1	W	16	30·005	30·077	30·092	30·149	48·9	35·4	E
17	30·136	30·234	30·230	30·228	50·9	43·9	E	17	30·163	30·181	30·115	30·047	42·9	35·8	ENE
18	30·141	30·096	29·996	29·989	68·3	42·9	S	18	29·948	29·917	29·880	29·885	51·3	41·7	S
19	29·933	29·905	29·854	29·912	73·7	47·3	ENE	19	29·882	29·875	29·819	29·854	67·4	47·6	S
20	29·935	29·988	29·977	30·076	80·3	47·7	E	20	29·905	29·975	29·987	30·140	69·1	50·7	N
21	30·091	30·121	30·085	30·124	75·0	47·8	E	21	30·180	30·191	30·150	30·168	53·1	43·3	NE
22	30·120	30·104	30·053	30·042	66·1	48·0	E	22	30·145	30·152	30·082	30·077	61·3	42·2	ENE
23	29·978	30·012	29·989	30·040	70·3	44·4	NNE	23	30·080	30·073	30·017	30·085	63·9	41·3	ENE
24	30·023	30·003	29·949	30·003	75·4	42·2	NE	24	30·093	30·088	30·031	30·068	70·4	41·3	NNE
25	30·001	30·001	29·961	30·008	73·9	43·8	ENE	25	30·074	30·089	30·073	30·093	61·7	45·7	E
26	29·999	30·004	29·953	30·003	68·4	44·8	NE	26	30·078	30·046	29·991	30·003	63·8	42·6	WNW
27	29·985	29·990	29·925	29·941	60·4	42·9	E	27	29·995	29·967	29·911	29·914	62·5	42·2	WNW
28	29·939	29·930	29·885	29·932	64·7	40·0	N	28	29·918	29·932	29·908	29·890	54·0	42·2	WNW
29	29·929	29·925	29·864	29·894	61·8	46·6	W	29	29·790	29·698	29·677	29·725	50·0	40·9	W
30	29·918	29·968	29·997	30·075	57·3	42·1	NW	30	29·744	29·786	29·835	29·863	50·5	38·9	NW

MAY, 1893.

KEW.								GLASGOW.							
	BAROMETER.				TEMPERATURE.		Direction of wind at noon		BAROMETER.				TEMPERATURE.		Direction of wind at noon
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.		Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	
1	30·063	30·074	30·025	30·023	60·1	42·0	WSW	1	29·884	29·881	29·862	29·872	51·2	36·8	W
2	30·005	30·009	29·933	29·956	64·3	44·7	SW	2	29·853	29·845	29·800	29·856	45·2	42·0	ENE
3	29·983	30·114	30·164	30·251	66·8	49·1	W	3	29·963	30·055	30·122	30·174	58.0	44·6	W
4	30·287	30·327	30·285	30·307	73·0	43·0	S	4	30·175	30·210	30·207	30·234	61·2	45·9	W
5	30·308	30·329	30·342	30·391	70·2	45·0	ENE	5	30·247	30·291	30·270	30·312	63·3	46·9	ENE
6	30·383	30·386	30·355	30·362	65·0	44·8	SE	6	30·329	30·328	30·277	30·297	63·1	45·0	NE
7	30·355	30·344	30·281	30·312	57·3	42·7	NE	7	30·348	30·380	30·378	30·454	59·3	43·3	ESE
8	30·302	30·300	30·248	30·257	63·1	41·8	NE	8	30·474	30·495	30·449	30·478	59·4	44·4	E
9	30·235	30·219	30·155	30·218	68·2	46·8	NE	9	30·473	30·465	30·393	30·427	64·5	44·1	ENE
10	30·220	30·227	30·197	30·282	68·6	45·9	NNE	10	30·420	30·388	30·329	30·324	64·2	42·7	SW
11	30·272	30·285	30·224	30·248	69·3	42·0	N	11	30·275	30·242	30·201	30·147	58·8	39·6	W
12	30·196	30·143	30·043	30·078	74·0	42·6	W	12	30·061	30·024	29·989	30 011	53·9	50·7	WSW
13	30·112	30·147	30·097	30·126	71·8	47·9	NNE	13	30·053	30·099	30·032	30·023	66·3	50·1	SW
14	30·086	30·027	29·905	29·893	72·8	49·0	ENE	14	30·010	30·014	29·936	29·951	70·2	50·1	W
15	29·843	29·838	29·822	29·855	76·2	49·9	E	15	29·959	29·978	29·947	29 949	59·0	47·6	NE
16	29·812	29·803	29·724	29·716	71·5	53·0	SE	16	29·885	29·839	29·757	29·727	53·7	44·7	NE
17	29·645	29·568	29·476	29·538	67·1	49·4	E	17	29·669	29·611	29·551	29·538	54·1	46·0	ENE
18	29·557	29·578	29·575	29·652	66·5	49·8	S	18	29·490	29·480	29·443	29·463	60·7	47·0	NE
19	29·604	29·651	29·626	29·638	65·1	51·3	S	19	29·470	29·506	29·476	29·450	61·1	51·1	S
20	29·635	29·652	29·608	29·634	65·0	51·4	S	20	29·426	29·430	29·457	29·487	60·8	50·1	W
21	29·652	29·694	29·703	29·766	65·2	49·8	SSW	21	29·500	29 554	29·622	29·698	57·1	49·9	SSE
22	29·800	29·857	29·887	29·950	69·8	50·7	W	22	29·756	29·818	29·798	29·775	64·7	49·1	SW
23	29·946	29·925	29·860	29·869	67·2	50·2	S	23	29·643	29·638	29·626	29·633	60·4	50·1	SW
24	29·878	29·895	29·939	30·026	64·8	48·1	W	24	29·600	29·728	29·866	29·953	61·1	49·9	W
25	30·081	30·099	30·057	30·139	70·3	47·2	W	25	29·901	29·920	30·022	30·115	59·4	51·0	W
26	30·148	30·186	30·188	30·202	60·6	50·2	NW	26	30·143	30·185	30·184	30·211	58·1	47·9	W
27	30·186	30·191	30·198	30·209	63·2	47·6	N	27	30·209	30·222	30·208	30·213	65·7	47·1	W
28	30·193	30·186	30·122	30·103	66·9	47·7	NW	28	30·188	30·182	30·118	30·110	65·1	53·1	W
29	30·066	30·037	29·985	30·035	70·9	52·2	NW	29	30·079	30·131	30·137	30·158	56·7	47·0	E
30	30·045	30·064	30·034	30·044	56·2	47·8	NNE	30	30·143	30·119	30·062	30·080	61·1	45·1	N
31	30·014	30·018	29·979	30·015	59·1	39·9	NW	31	30·044	30·040	29·992	30·009	61·6	43·0	W

JUNE, 1893.

1	30·015	29·992	29·941	29·942	62·6	38·9	NW	1	29·982	29·965	29·914	29·906	62·8	48·1	SW
2	29·915	29·904	29·875	29·905	63·8	47·2	E	2	29·875	29·862	29·799	29·816	65·5	42·9	W
3	29·887	29·914	29·926	30·006	67·6	39·8	NE	3	29·821	29·836	29·851	29·901	66·6	45·0	SW
4	30·067	30·151	30·180	30·243	69·8	51·9	W	4	29·934	29·997	30·050	30·125	62·8	50·1	SW
5	30·241	30·248	30·239	30·324	71·1	52·4	W	5	30·172	30·234	30·249	30·291	68·5	48·2	W
6	30·353	30·379	30·329	30·358	68·3	47·9	E	6	30·277	30·265	30·248	30·307	63·1	54·1	SE
7	30·353	30·359	30·288	30·310	71·0	51·7	E	7	30·330	30·356	30·338	30·349	66·9	54·0	W
8	30·307	30·315	30·235	30·266	70·5	51·2	NE	8	30·361	30·369	30·320	30·355	65·9	51·7	ENE
9	30·272	30·284	30·228	30·297	70·3	47·3	NNE	9	30·348	30·364	30·334	30·358	62·9	50·0	NE
10	30·305	30·337	30·282	30·286	67·1	51·7	NE	10	30·360	30·364	30·319	30·312	69·9	50·6	E
11	30·244	30·233	30·140	30·134	63·7	49·2	NE	11	30·277	30·243	30·157	30·152	73·3	51·7	NE
12	30·094	30·103	30·022	30·024	66·6	52·2	NE	12	30·142	30·116	30·042	30·060	73·2	50·2	E
13	29·974	29·939	29·900	29·904	76·1	54·4	E	13	30·058	30·044	29·975	29·975	68·8	51·2	E
14	29·846	29·825	29·778	29·838	79·7	55·6	ENE	14	29·975	29·940	29·886	29·954	71·0	51·9	NE
15	29·850	29·907	29·959	30·031	80·3	57·1	E	15	29·978	30·009	29·982	30·021	77·1	51·2	NE
16	30·067	30·107	30·096	30·178	82·6	55·4	N	16	30·039	30·081	30·125	30·198	74·7	57·2	SW
17	30·234	30·260	30·225	30·285	84·6	56·2	E	17	30·256	30·306	30·309	30·326	75·1	60·8	W
18	30·312	30·285	30·202	30·179	80·9	56·2	E	18	30·327	30·298	30·211	30·159	79·7	55·3	WSW
19	30·092	29·986	29·821	29·829	86·1	50·6	N	19	30·079	30·022	29·935	29·915	72·0	56·2	W
20	29·807	29·844	29·813	29·833	63·3	51·1	NE	20	29·890	29·878	29·849	29·851	62·3	52·8	W
21	29·835	29·838	29·801	29·783	68·3	50·5	SW	21	29·857	29·843	29·791	29 737	67·8	47·6	W
22	29·721	29·593	29·503	29·448	70·2	48·0	SSW	22	29·642	29·540	29·450	29·404	58·4	48·1	WSW
23	29·346	29·305	29·324	29·374	64·6	53·0	W	23	29·357	29·358	29·359	29·381	52·3	46·2	NW
24	29·356	29·474	29·578	29·665	61·6	50·5	W	24	29·408	29·448	29·462	29·535	61·9	49·4	W
25	29·708	29·774	29·806	29·842	66·7	48·8	WNW	25	29·584	29·630	29·683	29·720	60·2	47·0	W
26	29·816	29·794	29·746	29·760	67·3	47·7	E	26	29·732	29·741	29·692	29·714	64·9	44·1	WNW
27	29·651	29·642	29·646	29·660	70·3	57·8	SW	27	29·448	29·393	29·400	29·362	68·0	51·9	SW
28	29·601	29·638	29·684	29·786	71·8	57·1	SSW	28	29·285	29·298	29·332	29·453	67·0	56·7	S
29	29·825	29·937	30·001	30·089	69·5	53·9	SW	29	29·578	29·757	29·909	30·042	59·2	54·0	W
30	30·137	30·067	30·107	30·136	72·6	47·9	S	30	30·098	30·146	30·120	30·121	70·4	46·7	SW

JULY, 1893.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·122	30·100	30·058	30·094	68·6	61·0	E	1	30·111	30·084	30·030	30·053	75·2	52·4	E
2	30·092	30·101	30·060	30·086	83·3	56·9	ENE	2	30·064	30·094	30·086	30·126	75·7	54·6	W
3	30·089	30·087	30·058	30·080	78·8	57·0	E	3	30·144	30·178	30·164	30·178	67·7	54·3	E
4	30·030	29·987	29·828	29·827	71·6	54·9	E	4	30·144	30·118	30·050	30·050	67·8	54·0	E
5	29·738	29·756	29·797	29·888	73·8	54·4	N	5	30·023	30·046	30·042	30·082	62·9	51·9	NE
6	29·929	29·956	29·933	29·994	81·3	54·2	NE	6	30·068	30·048	29·996	29·972	66·0	51·8	NE
7	29·954	29·936	29·837	29·814	85·6	53·6	S	7	29·914	29·882	29·814	29·774	77·9	55·2	NE
8	29·724	29·706	29·687	29·687	80·9	59·9	SW	8	29·742	29·705	29·636	29·635	70·2	55·7	ENE
9	29·683	29·714	29·778	29·854	72·1	56·8	SSW	9	29·526	29·442	29·521	29·615	65·3	56·9	SE
10	29·864	29·865	29·830	29·796	73·8	57·1	S	10	29·689	29·725	29·731	29·763	70·1	55·9	ESE
11	29·719	29·673	29·609	29·590	74·5	57·4	S	11	29·751	29·754	29·719	29·750	69·4	55·1	E
12	29·518	29·525	29·513	29·559	69·5	57·2	N	12	29·726	29·741	29·734	29·729	60·8	54·7	E
13	29·576	29·640	29·693	29·762	64·0	57·9	N	13	29·714	29·732	29·740	29·790	58·8	51·3	SSE
14	29·788	29·843	29·854	29·930	63·2	50·3	NNE	14	29·834	29·884	29·893	29·928	64·1	49·6	NW
15	29·938	29·960	29·928	29·922	66·4	48·1	WNW	15	29·929	29·931	29·899	29·877	62·0	52·6	ENE
16	29·862	29·834	29·821	29·877	65·9	56·0	S	16	29·830	29·820	29·773	29·726	65·3	49·7	W
17	29·877	29·904	29·847	29·915	68·3	55·0	W	17	29·725	29·716	29·754	29·834	61·2	50·9	WNW
18	29·965	29·996	29·955	29·892	71·0	51·7	W	18	29·833	29·892	29·806	29·568	61·3	51·9	SW
19	29·774	29·702	29·655	29·598	69·2	60·8	SW	19	29·463	29·422	29·424	29·262	65·9	52·9	WSW
20	29·544	29·519	29·548	29·596	71·3	61·8	WSW	20	29·274	29·354	29·402	29·504	61·7	52·8	W
21	29·634	29·499	29·714	29·738	71·3	56·6	SW	21	29·561	29·624	29·649	29·680	64·0	50·1	NW
22	29·775	29·896	29·977	30·091	70·7	55·2	W	22	29·707	29·794	29·909	29·994	61·3	48·2	NW
23	30·135	30·142	30·097	30·002	71·2	50·2	SSW	23	29·994	29·976	29·902	29·842	68·6	44·1	S
24	29·901	29·879	29·817	29·818	73·9	60·2	SW	24	29·726	29·684	29·606	29·582	63·5	53·4	SW
25	29·795	29·825	29·814	29·842	69·1	58·7	W	25	29·564	29·542	29·619	29·670	62·3	51·7	W
26	29·813	29·820	29·870	29·967	69·0	55·4	NNW	26	29·698	29·819	29·928	30·034	64·0	51·7	NW
27	30·039	30·122	30·155	30·229	67·4	54·7	N	27	30·118	30·170	30·170	30·202	67·9	47·8	W
28	30·242	30·258	30·184	30·174	74·0	48·9	NW	28	30·174	30·154	30·131	30·130	63·1	50·4	SW
29	30·104	30·065	29·983	29·912	68·9	58·9	SW	29	30·052	29·969	29·914	29·871	60·9	53·3	WSW
30	29·846	29·833	29·835	29·883	66·6	54·1	WNW	30	29·855	29·852	29·854	29·876	63·6	49·3	WNW
31	29·851	29·851	29·864	29·943	67·1	50·9	NW	31	29·855	29·876	29·874	29·929	65·2	49·8	NW

AUGUST, 1893.

1	29·954	29·962	29·907	29·900	67·8	49·5	NW	1	29·891	29·880	29·834	29·782	59·1	47·1	W
2	29·799	29·836	29·853	29·913	69·4	56·3	NW	2	29·758	29·759	29·733	29·654	62·9	51·3	W
3	29·864	29·829	29·793	29·768	68·5	55·9	SW	3	29·493	29·433	29·482	29·498	63·4	52·9	W
4	29·699	29·692	29·683	29·704	69·4	55·8	SW	4	29·478	29·468	29·477	29·556	63·1	53·0	SSW
5	29·742	29·834	29·895	30·007	67·7	52·2	WNW	5	29·658	29·806	29·913	29·998	61·8	51·4	N
6	30·055	30·101	30·056	30·064	69·3	46·3	SSW	6	30·019	29·996	29·912	29·840	62·0	47·6	SW
7	30·018	30·050	30·032	30·079	76·3	57·3	SW	7	29·784	29·808	29·851	29·958	67·9	53·6	S
8	30·085	30·109	30·081	30·100	80·7	53·3	SSE	8	29·981	30·043	30·071	30·086	67·2	57·7	S
9	30·055	30·032	29·951	29·926	82·8	61·8	ESE	9	30·072	30·065	29·989	29·997	80·1	58·7	E
10	29·859	29·912	29·923	29·983	79·2	63·2	NW	10	29·960	29·932	29·871	29·919	69·5	57·6	E
11	30·020	30·063	30·075	30·121	78·6	62·2	WSW	11	29·946	29·965	29·919	29·952	67·4	58·3	S
12	30·114	30·136	30·105	30·128	77·6	63·8	SW	12	30·004	30·075	30·114	30·155	69·8	56·2	W
13	30·121	30·136	30·117	30·150	78·8	61·0	E	13	30·144	30·143	30·108	30·143	74·1	51·0	WNW
14	30·150	30·184	30·150	30·184	82·1	60·8	S	14	30·144	30·170	30·138	30·162	77·7	59·7	W
15	30·173	30·184	30·120	30·134	85·4	58·5	SSW	15	30·134	30·129	30·060	30·038	78·1	59·7	WSW
16	30·112	30·095	30·025	30·052	86·7	57·6	WSW	16	30·003	30·034	30·015	30·010	69·1	60·0	W
17	30·036	30·041	29·968	29·923	88·6	60·0	SW	17	29·942	29·920	29·884	29·822	72·0	59·2	SW
18	29·819	29·766	29·762	29·826	88·3	65·9	S	18	29·708	29·644	29·638	29·670	71·7	59·7	S
19	29·842	29·889	29·872	29·899	76·7	62·3	S	19	29·670	29·690	29·648	29 573	69·1	56·6	SW
20	29·862	29·802	29·666	29·744	75·9	59·1	SW	20	29·503	29·502	29·506	29·410	66·5	56·8	SW
21	29·643	29·602	29·659	29·789	74·5	58·6	SW	21	29·201	29·163	29·179	29·344	61·3	55·0	S
22	29·815	29·822	29·807	29·913	72·4	55·7	SSW	22	29·444	29·538	29·569	29·684	61·3	54·7	SW
23	29·927	29·934	29·912	29·959	67·5	55·4	WNW	23	29·713	29·721	29·744	29·768	62·7	51·5	SW
24	29·991	30·036	30·064	30·166	69·8	53·1	W	24	29·713	29·819	29·931	30·022	62·5	49·8	WNW
25	30·197	30·241	30·206	30·245	69·2	52·1	NW	25	30·074	30·137	30·171	30·231	61·7	51·0	W
26	30·241	30·230	30·170	30·168	60·0	48·1	NNW	26	30·210	30·214	30·194	30·167	62·5	52·2	W
27	30·123	30·115	30·106	30·167	62·7	48·9	N	27	30·158	30·214	30·221	30·275	60·5	52·0	NE
28	30·190	30·245	30·240	30·291	66·9	48·0	NE	28	30·310	30·323	30·306	30·326	62·2	42·3	W
29	30·294	30·288	30·225	30·209	66·5	43·6	NNE	29	30·318	30·316	30·256	30·236	66·1	46·7	W
30	30·158	30·132	30·029	30·025	71·8	57·4	N	30	30·172	30·125	30·074	30·035	65·2	51·4	WNW
31	29·967	29·959	29·895	29·937	70·1	54·2	NW	31	29·955	29·933	29·934	29·962	63·7	51·3	NE

SEPTEMBER, 1893.

KEW.								GLASGOW.							
	Barometer.				Temperature.		Direction of wind at noon		Barometer.				Temperature.		Direction of wind at noon
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.		Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	
1	29·928	30·012	30·032	30·050	66·1	56·4	NE	1	29·942	29·962	29·966	30·003	62·4	51·1	WNW
2	30·041	30·087	30·093	30·191	69·2	56·6	NW	2	30·014	30·090	30·117	30·192	67·9	55·9	WNW
3	30·218	30·261	30·236	30·243	68·1	53·4	NE	3	30·206	30·244	30·211	30·200	64·3	50·9	ESE
4	30·231	30·214	30·122	30·099	71·5	48·0	SE	4	30·160	30·138	30·065	30·034	67·7	50·0	SW
5	30·029	29·992	29·868	29·864	77·1	45·0	SE	5	29·962	29·910	29·804	29·750	65·8	52·2	WSW
6	29·814	29·813	29·730	29·725	77·2	51·6	WSW	6	29·732	29·700	29·626	29·540	62·2	52·0	SW
7	29·678	29·727	29·786	29·846	73·0	55·2	W	7	29·476	29·543	29·582	29·575	63·8	53·2	W
8	29·782	29·716	29·654	29·650	67·6	53·8	WNW	8	29·513	29·491	29·565	29·751	60·2	49·7	WNW
9	29·756	29·883	29·872	29·889	63·6	49·6	NNW	9	29·806	29·855	29·863	29·949	56·0	43·1	WNW
10	29·894	29·967	29·986	30·092	61·9	48·0	NE	10	29·994	30·082	30·115	30·194	56·1	38·0	E
11	30·113	30·187	30·172	30·258	65·2	47·4	E	11	30·234	30·278	30·265	30·287	58·0	36·7	W
12	30·245	30·277	30·209	30·242	66·2	46·8	ENE	12	30·276	30·256	30·198	30·139	59·8	38·6	SW
13	30·211	30·193	30·102	30·116	67·6	41·9	W	13	29·979	29·826	29·818	29·967	61·4	52·1	SW
14	30·151	30·222	30·206	30·236	74·9	53·9	WNW	14	30·009	30·067	30·090	30·048	64·9	54·3	WSW
15	30·207	30·175	30·030	29·990	73·8	47·8	W	15	29·931	29·906	29·874	29·836	63·9	54·9	WSW
16	29·911	29·867	29·761	29·720	65·6	46·0	NE	16	29·814	29·824	29·798	29·788	60·3	48·3	W
17	29·685	29·739	29·721	29·761	63·4	53·2	N	17	29·707	29·633	29·550	29·554	57·9	47·8	W
18	29·700	29·670	29·605	29·589	66·2	54·0	SW	18	29·504	29·402	29·358	29·278	58·6	53·7	WSW
19	29·518	29·500	29·432	29·419	67·0	58·0	SW	19	29·098	29·166	29·132	29·141	56·8	46·1	WSW
20	29·394	29·437	29·410	29·439	61·1	48·7	WSW	20	29·213	29·305	29·328	29·382	54·2	43·0	WNW
21	29·515	29·521	29·414	29·484	52·9	44·4	SW	21	29·398	29·421	29·470	29·488	49·2	38·7	NNW
22	29·537	29·635	29·676	29·712	60·1	42·0	SW	22	29·508	29·588	29·584	29·434	51·4	40·3	NNW
23	29·630	29·572	29·573	29·744	52·0	38·9	NW	23	29·361	29·471	29·554	29·692	50·0	33·7	NW
24	29·834	29·973	29·995	30·050	58·2	38·7	NW	24	29·820	29·902	29·844	29·748	52·1	41·2	W
25	29·985	29·974	29·974	30·006	58·7	40·9	WNW	25	29·809	29·942	29·945	29·931	49·6	41·8	W
26	29·975	29·961	29·890	29·880	62·1	46·8	SW	26	29·852	29·925	29·788	29·765	48·9	41·8	E
27	29·767	29·776	29·744	29·827	64·4	52·4	WSW	27	29·702	29·652	29·621	29·626	55·1	46·2	NW
28	29·840	29·803	29·656	29·479	61·2	49·5	SW	28	29·521	29·302	29·218	29·087	58·0	48·1	SW
29	29·325	29·344	29·338	29·333	63·1	53·7	SW	29	28·938	28·871	28·919	28·937	53·8	47·3	SW
30	29·255	29·263	29·346	29·465	61·4	49·0	W	30	28·955	28·992	29·087	29·216	55·3	46·6	WSW

OCTOBER, 1893.

1	29·529	29·588	29·606	29·664	61·9	40·4	S	1	29·316	29·410	29·482	29·515	54·8	45·8	W
2	29·633	29·603	29·452	29·358	60·0	45·5	SSW	2	29·480	29·404	29·282	29·230	53·7	44·1	SSW
3	29·317	29·371	29·294	29·266	57·6	44·1	WSW	3	29·190	29·183	29·134	29·038	52·5	41·2	SW
4	29·205	29·185	29·171	29·206	58·1	43·7	WSW	4	28·970	28·984	28·976	29·014	47·0	39·3	S
5	29·217	29·280	29·276	29·304	58·9	46·0	SW	5	29·040	29·086	29·115	29·176	52·2	31·9	W
6	29·323	29·490	29·551	29·600	60·0	48·0	SW	6	29·194	29·281	29·371	29·471	53·5	42·2	W
7	29·530	29·574	29·530	29·504	60·5	44·7	SSW	7	29·504	29·559	29·557	29·598	52·5	44·2	S
8	29·581	29·704	29·730	29·792	61·0	43·1	SW	8	29·574	29·528	29·582	29·704	48·6	42·5	W
9	29·755	29·684	29·585	29·555	60·4	41·4	ENE	9	29·761	29·782	29·699	29·671	53·0	38·8	ENE
10	29·667	29·853	29·923	29·892	57·4	51·0	WSW	10	29·612	29·627	29·647	29·649	52·4	40·3	W
11	29·763	29·796	29·798	29·840	60·9	50·0	WSW	11	29·584	29·558	29·609	29·709	53·4	40·9	SW
12	29·884	30·041	30·108	30·189	55·7	43·5	NW	12	29·754	29·849	29·898	29·951	52·1	42·6	W
13	30·168	30·198	30·149	30·085	55·5	39·0	W	13	29·992	30·005	29·896	29·699	51·3	40·2	SW
14	29·929	29·870	29·872	29·922	64·0	54·0	W	14	29·593	29·581	29·626	29·643	58·3	49·3	SW
15	29·922	29·950	29·943	29·941	64·0	58·4	SW	15	29·604	29·613	29·604	29·622	59·7	55·6	WSW
16	29·931	29·943	29·966	30·046	65·3	56·4	SW	16	29·593	29·649	29·724	29·809	58·2	51·4	SW
17	30·037	30·055	29·996	29·967	60·0	53·0	WSW	17	29·871	29·928	29·934	29·994	55·5	44·6	WNW
18	29·949	30·099	30·198	30·291	57·3	44·0	N	18	30·039	30·080	30·117	30·176	53·3	38·2	SW
19	30·325	30·379	30·352	30·378	56·9	38·7	NNE	19	30·192	30·234	30·248	30·267	55·0	49·3	WSW
20	30·350	30·385	30·324	30·321	59·2	44·9	SW	20	30·217	30·138	30·054	30·004	58·1	49·5	SSW
21	30·283	30·261	30·207	30·190	63·7	52·3	SW	21	29·932	29·908	29·802	30·063	59·6	46·0	SW
22	30·169	30·196	30·220	30·301	55·6	43·1	SW	22	30·053	30·047	30·070	30·170	53·8	44·4	WSW
23	30·342	30·425	30·426	30·444	55·5	38·9	N	23	30·254	30·356	30·316	30·245	54·1	46·9	W
24	30·405	30·407	30·315	30·257	55·4	45·1	SW	24	30·099	30·092	29·969	29·814	53·0	49·9	SW
25	30·153	30·056	29·905	29·792	55·9	47·3	SW	25	29·617	29·532	29·514	29·448	52·9	40·0	W
26	29·793	29·830	29·777	29·765	52·6	45·0	W	26	29·394	29·521	29·520	29·618	43·5	35·7	SW
27	29·840	29·974	30·014	30·048	52·1	39·7	WNW	27	29·728	29·804	29·725	29·693	50·7	39·3	SW
28	29·988	29·930	29·816	29·767	57·7	40·9	SW	28	29·570	29·412	29·424	29·519	53·3	43·7	SW
29	29·727	29·778	29·744	29·809	56·2	44·9	W	29	29·466	29·478	29·584	29·719	45·3	35·7	W
30	29·816	29·875	29·899	29·993	49·9	36·2	NNW	30	29·810	29 901	29·950	30·069	43·3	31·0	W
31	30·034	30·097	30·117	30·141	43·3	30·4	N	31	30·075	30·155	30·062	29·855	40·0	28·7	W

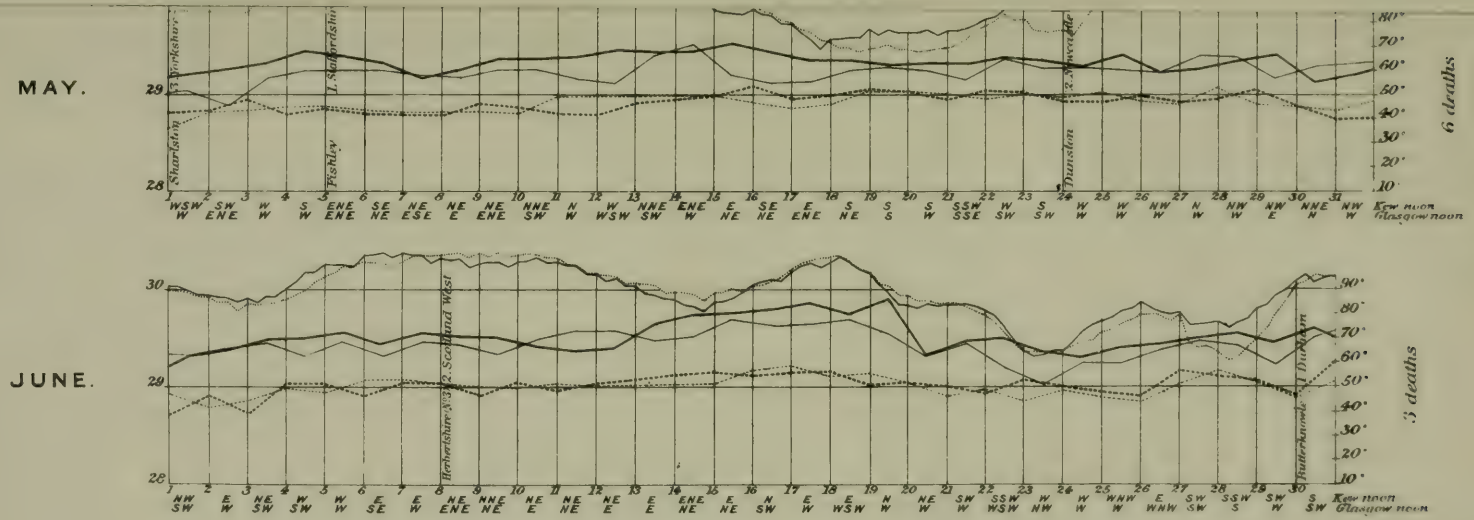
NOVEMBER, 1893.

KEW.								GLASGOW.							
	BAROMETER.				TEMPERATURE.				BAROMETER.				TEMPERATURE.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·081	29·972	29·777	29·765	49·1	28·9	SW	1	29·541	29·476	29·482	29·521	46·5	35·7	WSW
2	29·754	29·654	29·719	29·847	51·0	45·0	SW	2	29·492	29·491	29·582	29·690	46·4	36·8	W
3	29·897	29·901	29·880	29·855	59·6	43·0	SW	3	29·708	29·646	29·527	29·496	42·4	39·3	E
4	29·811	29·861	29·969	30·034	55·0	39 8	N	4	29·787	29·973	30·019	29·991	42·3	31·7	W
5	29·966	29·908	29·805	29·843	43·2	34·2	NNE	5	29·759	29·710	29·278	30·009	42·1	30·8	W
6	29·903	29·996	30·038	30·160	46·8	32·8	N	6	30·110	30·224	30·256	30·352	42·8	32·7	NNW
7	30·172	30·227	30·238	30·274	45·1	33·9	N	7	30·382	30·433	30·440	30·457	41·9	31·9	ENE
8	30·250	30·282	30·244	30·272	44·7	33·2	NE	8	30·428	30·441	30·431	30·479	46·9	33·9	NNE
9	30·239	30·246	30·169	30·194	43·3	38·4	NE	9	30·482	30·492	30·459	30·474	43·8	35·1	NE
10	30·175	30·220	30·211	30·259	45·1	37·9	NE	10	30·455	30·469	30·452	30·466	43·0	39·3	ENE
11	30·246	30·299	30·305	30·355	49·1	42·2	NE	11	30·461	30·451	30·470	30·483	46·2	34·9	N
12	30·325	30·346	30·278	30·244	44·1	37·8	E	12	30·452	30·438	30·380	30·325	44·5	41·7	E
13	30·146	30·072	29·970	29·937	43·2	30·6	E	13	30·240	30·165	30·021	29·951	43·9	35·4	NE
14	29·880	29·828	29·722	29·631	42·8	31·0	E	14	29·851	29·818	29·762	29·798	41·2	32·9	S
15	29 626	29·743	29·832	29·937	45·3	38·6	N	15	29·826	29·894	29·902	29·920	41·1	30·1	NNE
16	29·925	29·881	29·694	29·480	48·3	41·1	ESE	16	29·842	29·722	29·484	29·206	46·7	39·3	ESE
17	29·231	29·138	28·995	29·056	57·2	40·0	SSW	17	28·760	28·526	28·748	28·936	51·4	40·7	WSW
18	28·969	29·063	29·225	29·391	42·2	31·7	NNW	18	29·211	29·610	29·792	29·889	43·4	30·9	N
19	29·519	29·594	29·615	29·742	42·0	31·8	N	19	29·935	30·046	30·127	30·232	36·8	29·9	N
20	29·838	29·953	30·076	30·219	43·1	39·1	N	20	30·306	30·379	30·429	30·500	40·1	34·2	N
21	30·322	30·415	30·419	30·422	44·7	39·1	N	21	30·518	30·519	30·407	30·218	43·4	26·9	WSW
22	30·304	30·138	29·962	29·977	44·8	34·0	W	22	30·010	30·017	30·070	30·126	44·5	32·9	NNW
23	29·944	30·056	30·084	30·145	38·3	30·6	NW	23	30·142	30·224	30·118	30·019	43·2	29·8	SW
24	30·085	30·108	30·042	30·014	45·1	30·1	W	24	30·000	29·946	29·795	29·733	47·7	42·7	SW
25	29·939	29·882	29·677	29·455	49·8	41·4	SW	25	29·622	29·440	29·277	29·279	51·0	43·7	WSW
26	29·354	29·510	29·758	30·033	50·3	34·2	N	26	29·498	29·804	29·985	30·099	44·4	35·0	N
27	30·231	30·352	30·329	30·301	44·5	29·7	W	27	30·073	30·030	29·941	29·894	50·8	33·0	SW
28	30·251	30·285	30·296	30·326	52·5	44·3	SW	28	29·884	29·960	29·922	29·908	54·0	50·1	WSW
29	30·302	30·286	30·161	30·102	50·5	46·7	SW	29	29·823	29·758	29·638	29·608	53·7	48·8	SW
30	29·977	29·898	29·784	29·728	50·8	45·4	SW	30	29·618	29·696	29·711	29·775	49·0	34·1	W

DECEMBER, 1893.

1	29·634	29·749	29·919	30·141	46·4	28·4	N	1	29·769	29·991	30·090	30·184	34·5	25·8	NW
2	30·296	30·416	30·429	30·458	35·6	24·4	N	2	30·306	30·370	30·269	30·198	43·7	24·6	W
3	30·413	30·400	30·297	30·271	36·2	20·8	NW	3	30·129	30·119	30·056	30·120	49·2	42·9	W
4	30·204	30·187	30·125	30·199	45·7	35·3	W	4	30·123	30·146	30·175	30·234	50·4	44·7	WNW
5	30·261	30·348	30·303	30·280	42·2	36·0	N	5	30·228	30·181	29·991	29·910	47·7	42·2	SW
6	30·180	30·132	30·065	29·911	48·2	37·5	SW	6	29·809	29·655	29·411	29·358	50·1	44·1	SW
7	29·722	29·724	29·747	29·784	48·0	37·1	W	7	29·309	29·319	29·377	29·356	45·8	33·9	W
8	29·604	29·419	29·347	29·266	49·1	36·2	SW	8	28·961	28·524	28·716	28·799	46·1	36·9	SW
9	29·199	29·361	29·437	29·556	48·6	34·2	W	9	28·861	28·988	29·127	29·260	41·1	33·0	WSW
10	29·560	29·436	29·271	29·439	47·0	40·7	S	10	29·271	29·130	28·680	28·923	43·8	33·4	E
11	29·387	29·430	29·505	29·573	47·1	31·1	W	11	29·146	29·234	29·292	29·356	41·8	35·1	SW
12	29·570	29·487	29·182	29·328	48·1	30·7	S	12	29·234	29·100	29·090	29·096	40·7	32·2	SSW
13	29·174	29·155	29·088	29·196	55·8	41·4	SW	13	28·974	28·732	28·729	29·109	38·8	32·9	ENE
14	29·422	29·768	30·012	30·236	46·9	33·7	N	14	29·247	29·640	29·830	29·898	41·2	33·8	W
15	30·359	30·395	30·410	30·472	49·9	31·3	SW	15	29·794	29·958	29·914	29·947	51·7	37·1	SW
16	30·481	30·515	30·456	30·418	50·7	41·1	SW	16	30·034	30·062	30·002	30·026	52·1	49·2	S
17	30·367	30·355	30·261	30·181	42·4	29·7	S	17	30·004	30·070	30·046	29·959	49·8	44·7	SSW
18	30·092	30·071	29·992	29·929	45·3	37·0	SSW	18	29·778	29·709	29·654	29·615	45·3	39·9	SW
19	29·736	29·473	29·159	29·148	47·7	41·9	S	19	29·379	29·026	28·781	28·652	44·8	37·9	SE
20	29·206	28·835	28·571	29·027	49·1	36·8	SSW	20	28·728	28·793	28·828	28·896	43·4	33·3	W
21	29·300	29·499	29·584	29·750	43·2	35·9	W	21	28·940	29·041	29·137	29·362	43·1	33·0	SSW
22	29·833	29·813	29·665	29·876	50·4	34·7	SSW	22	29·415	29·202	29·328	29·514	48·6	39·1	SW
23	30·014	30·145	30·199	30·244	46·3	35·8	WSW	23	29·640	29·805	29·890	29·959	47·1	40·9	S
24	30·222	30·202	30·076	29·941	46·9	40·4	S	24	29·888	29·814	29·576	29·495	49·3	43·4	S
25	29·950	30·147	30·207	30·253	48·0	40·6	W	25	29·564	29·763	29·773	30·036	43·8	38·6	SW
26	30·343	30·419	30·404	30·413	45·8	35·9	WSW	26	30·151	30·201	30·172	30·165	45·5	38·0	S
27	30·406	30·432	30·437	30·478	45·9	34·9	SW	27	30·130	30·142	30·158	30·234	48·3	44·2	SSE
28	30·526	30·586	30·598	30·649	41·5	30·9	SSW	28	30·296	30·390	30·428	30·486	48·7	47·1	S
29	30·653	30·689	30·688	30·725	40·8	31·9	SSW	29	30·509	30·578	30·575	30·604	47·3	43·7	SW
30	30·720	30·728	30·654	30·646	34·0	24·9	SE	30	30·571	30·574	30·541	30·534	48·2	43·5	WSW
31	30·570	30·528	30·412	30·367	32·9	22·8	SW	31	30·449	30·423	30·361	30·435	48·5	41·0	WSW

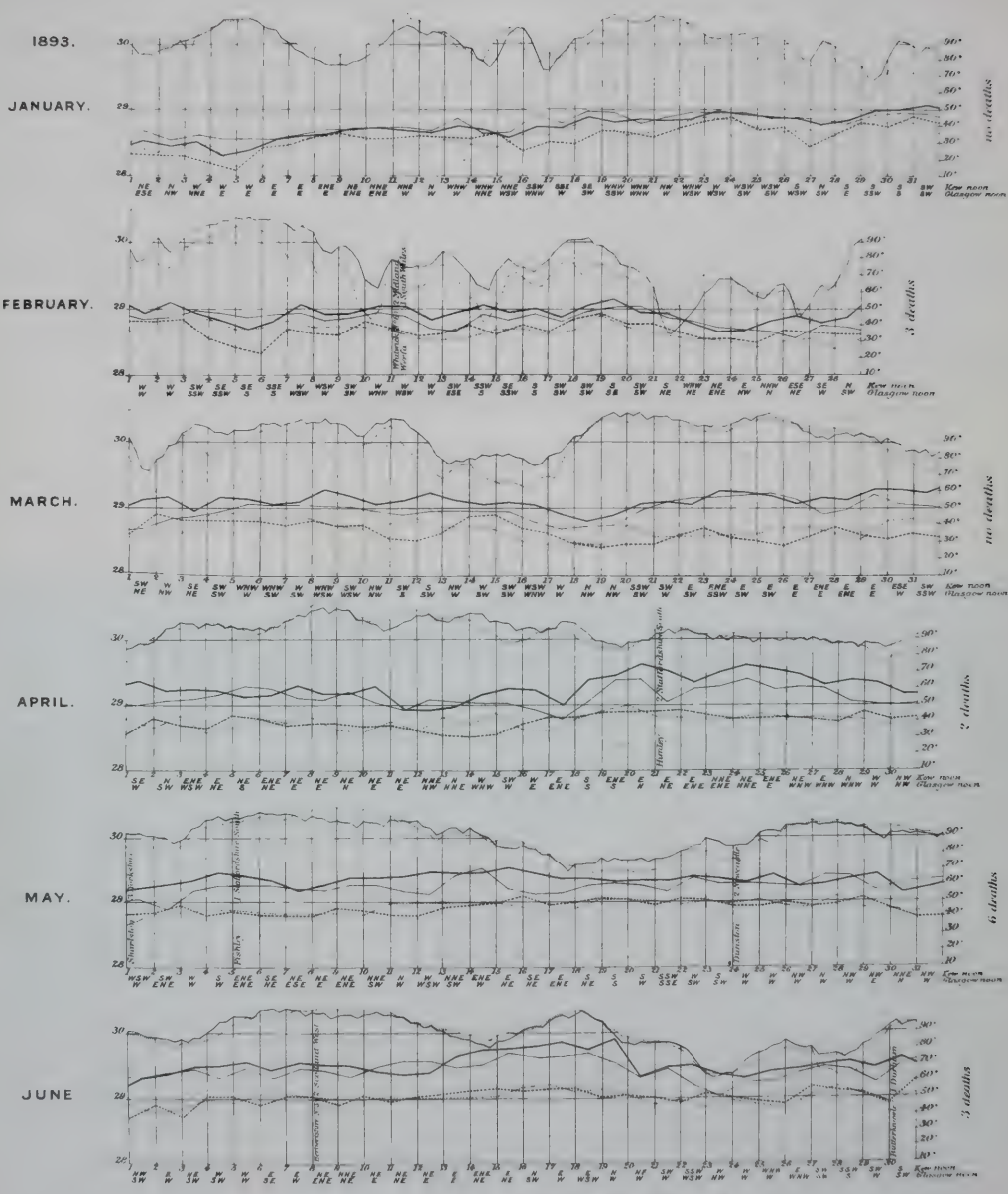
DIAGRAM SHEWING THE HEIGHT OF THE BAROMETER, THE MAXIMA & MINIMA TEMPERATURES & THE DIRECTION OF THE WIND AT THE OBSERVATORIES OF KEW & GLASGOW TOGETHER WITH THE EXPLOSIONS OF FIREDAMP IN ENGLAND & SCOTLAND.



The firm top line is the Barometer reading at Kew, taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The dotted top line is the Barometer reading at Glasgow taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The lower lines are the Maxima & Minima Temperatures at Kew & Glasgow, observed respectively at midnight & 9 p.m.
Kew lines are thick & Glasgow lines are thin.
The figures attached to the districts, shew the Number of Deaths caused by the Explosions.

And.w Reid & Comp.y Ltd. Newcastle upon Tyne

DIAGRAM SHEWING THE HEIGHT OF THE BAROMETER, THE MAXIMA & MINIMA TEMPERATURES & THE DIRECTION OF THE WIND AT THE OBSERVATORIES OF KEW & GLASGOW TOGETHER WITH THE EXPLOSIONS OF FIREDAMP IN ENGLAND & SCOTLAND



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Federated Institution of Mining Engineers.
Transactions 1893-4.

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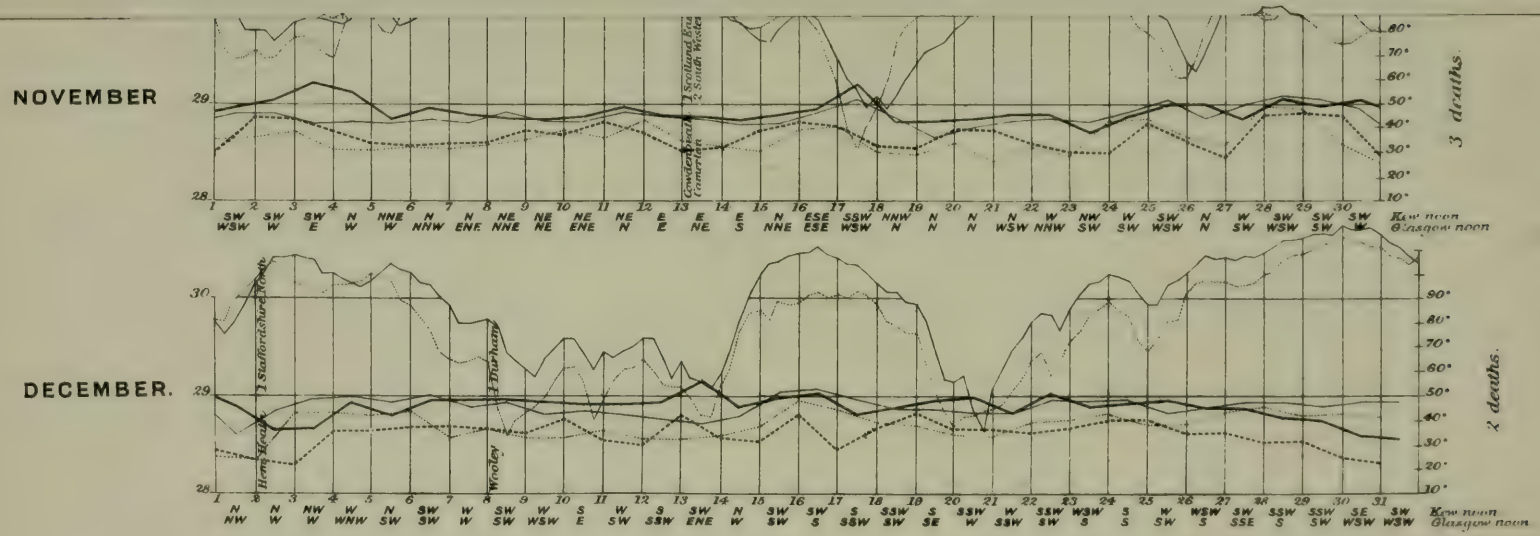

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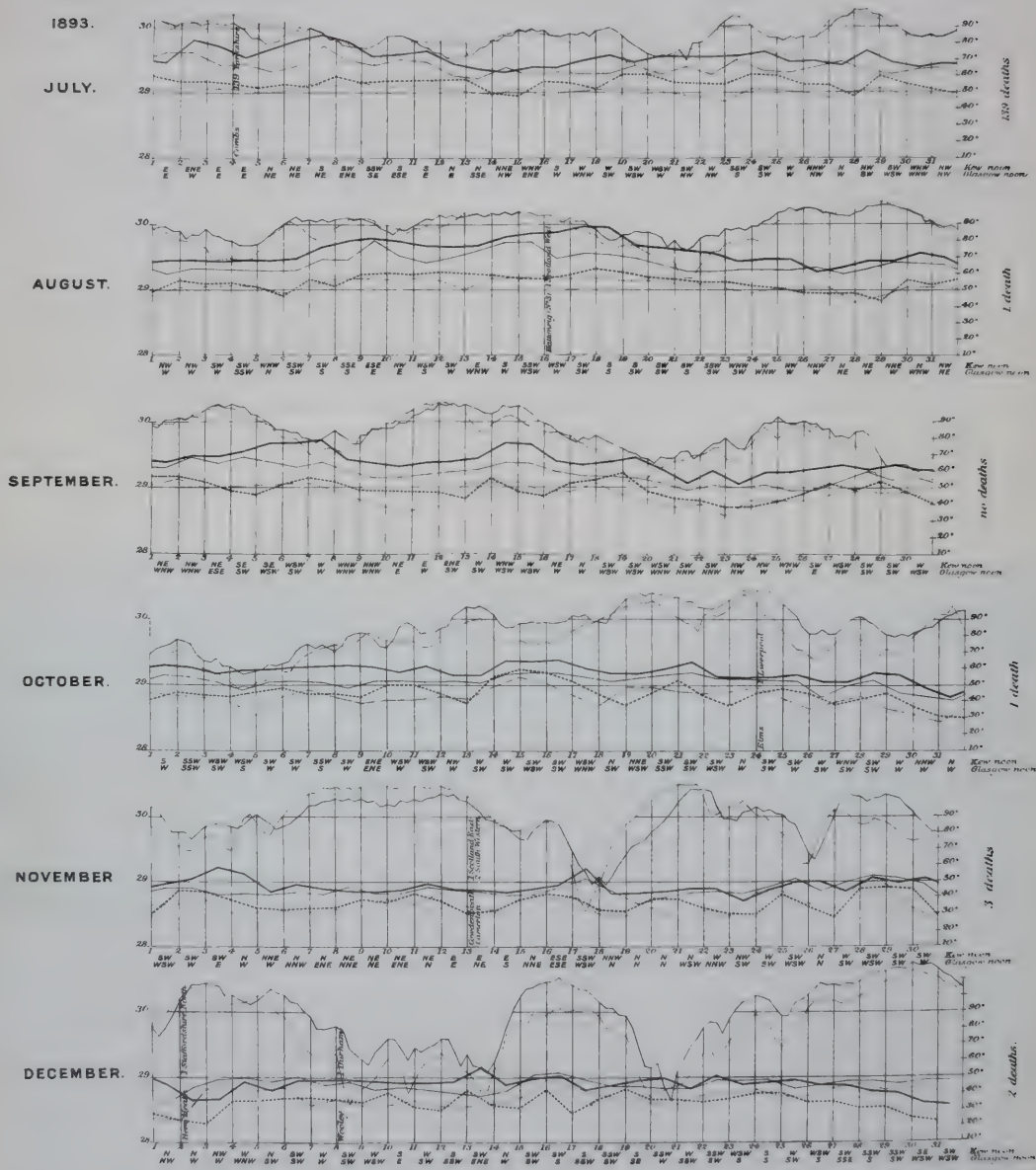
North of England Institute of Mining & Mechanical Engineers
Transactions, 1893-4

DIAGRAM SHEWING THE HEIGHT OF THE BAROMETER, THE MAXIMA & MINIMA TEMPERATURES & THE DIRECTION OF THE WIND AT THE OBSERVATORIES OF KEW & GLASGOW TOGETHER WITH THE EXPLOSIONS OF FIREDAMP IN ENGLAND & SCOTLAND.

1893.



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The dotted top line is the Barometer reading at Glasgow taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The lower lines are the Maxima & Minima Temperatures at Kew & Glasgow, observed respectively at midnight & 2 p.m.
Kew lines are thick & Glasgow lines are thin.
The figures attached to the districts, shew the Number of Deaths caused by the Explosions.

APPENDICES.

I.—NOTES OF PAPERS ON THE WORKING OF MINES. METALLURGY, ETC., FROM THE TRANSACTIONS OF FOREIGN SOCIETIES AND FOREIGN PUBLICATIONS.

THE ANTIMONY MINE OF FREYCENET, UPPER LOIRE, FRANCE.

La Mine d'Antimoine de Freycenet. *By* P. L. BURTHE. *Annales des Mines*, 1893, *series* 9, *vol. iv.*, *pages* 15-33.

The mining concession of Freycenet, covering an area of 570 acres, is situated 3¾ miles west of the little market-town of Lavoute-Chilhac. Within the concession the so-called plateau de Freycenet and plateau de Mont Rome are divided by the deep ravine of La Rhode from the broad plateau of the same name. West of this ravine the rocks are gneiss, east of it they are mica-schists. The gneiss is very felspathic, soft, more massive, and more regularly-bedded than the mica-schists; it appears to strike east-and-west. The mica-schists are very variable and thinly bedded, with a steep dip; their east-and-west strike on the plateau de Freycenet suddenly changes to a north-and-south strike on the plateau de Mont Rome. They are frequently faulted, but the faulting does not seem to extend much in depth. There are two varieties of mica-schists here which seem to have been unfavourable to the formation of metalliferous veins: a hard, reddish-white, very felspathic, slightly micaceous rock, and an extremely soft schist, almost entirely made up of flakes of bronze-coloured mica. A few veins of granulite are met with, but do not appear to have had any influence on the occurrence of the metalliferous veins, and the same statement applies to the lumps of green hornblendic rock occasionally found in contact with the antimony ore.

There are seven antimony-bearing veins (these alone are worked), three holding complex antimony-lead and argentiferous ores, and one of argentiferous galena. The antimony ore is a very pure sulphide, slightly altered here and there at the outcrop to a white or yellow oxide. The veins are either filled up with the country schist, or with a very compact bluish quartz, spangled with fine needles of stibnite when the vein is poor, and disappearing almost entirely as the vein becomes richer. Iron pyrites is not particularly abundant; pink or white calcite sometimes occurs, and there are traces of zinc blende, but no lead or arsenic; nor are the antimony veins ever found to be auriferous or argentiferous. The rich portions of a vein form little masses of practically pure ore, separated from one another by lengths very poor in ore or entirely barren. In this connexion veins No. 1 and No. 2 of the plateau de Freycenet are typical. They strike east-and-west; No. 1 for 196 feet in horizontal length has been worked uninterruptedly to a depth of 164 feet, and in No. 2 for 206 feet in horizontal length has been worked to a depth of 321 feet.

Desultory efforts were made to work the mine until it was re-opened, new shafts and adits being driven by its present owners. The production from the two veins just mentiond in nineteen months from June, 1889, to December, 1890, amounted to close upon 638 tons of marketable ore, which, being sold at the average price of £9 3s. 10d. per ton, yielded a gross profit of £2 10s. 9d. per ton.

The nature of the ores and the conditions of their occurrence as described for the Freycenet mining concession are repeated in similar concessions along the left bank of the Allier (Barlet, Chazelles, Lubilhac), and in other localities for which there is at present no concessions, but where old heaps of loose *débris* testify to the existence in bygone days of a far more extensive mining industry in that district. O. S. E.

THE ANALYSIS OF COAL.

Technische Analysen und Proben aus dem chemischen Laboratorium der k.k. geologischen Reichsanstalt. *By* C. VON JOHN *and* H. B. VON FOULLON. *Jahrbuch der k.k. geologischen Reichsanstalt, Vienna*, 1892, *vol. xlii., pages* 155-178.

The authors, in communicating determinations of the water, ash, and sulphur-content, and of the calorific power of coals from various formations and localities in the Austro-Hungarian empire, draw attention to the untrustworthiness of the Berthier method of determination. They state that by this method, with coals rich in hydrogen, the figures for the calorific power always come out too small. The report also contains chemical analyses of various Austro-Hungarian coals, cokes, patent fuel, and of an anthracite found at Orsilia, near Genoa (carbon 72·46 per cent. and ash 21·5 per cent.). O. S. E.

THE MANUFACTURE COLLIERY EXPLOSION, FRANCE.

L'Explosion de Grisou survenue au Puits de la Manufacture (Loire), le 6 Décembre, 1891. ANON. *Annales des Mines*, 1893, *series* 9, *vol. iv., pages* 235-251.

An explosion of fire-damp occurred at the Manufacture colliery, Loire, France, on December 6th, 1891, and resulted in the death of 62 persons, and in severe injury to 10 others. The explosion is attributed to the temporary stoppage of the ventilator on the morning of December 6th. During this stoppage inflammable gases escaped from old workings, and when the fan was restarted these gases (becoming mixed with fresh air) formed an explosive mixture. The mine was on fire, and one of the stoppings having become reduced in thickness in its upper part to a few inches the explosive mixture passed through and was ignited at the fire.

This accident illustrates the importance of isolating the working parts of the mine from old workings, and shows that the ventilating currents traversing old workings should be completely isolated from those passing through the working-places. It further proves that it is impossible to rely upon the efficiency of stoppings, and shows the necessity of passing a current of fresh air along them, in the case of a mine subject to spontaneous combustion. O. S. E.

THE HARDY FIRE-DAMP DECTECTOR.

Sur l'application des vibrations sonores à l'analyse des mélanges de deux gaz de densités différentes. *By* E. HARDY. *Comptes-Rendus hebdomadaires des Séances de l'Académie des Sciences*, 1893, *vol. cxvii., pages* 573-574.

When two organ-pipes, giving the same note, are sounded together by the aid of two distinct organ-bellows, filled with fresh air, a single note is obtained. If, however, one of the bellows, instead of being supplied with pure air, is supplied with a mixture of air and some other gas, the note of the organ-pipe is correspondingly modified, and the two pipes sounding together give more or less frequent beats according as the air is mixed more or less with the other gas.

The Hardy fire-damp detector consists of two separate bellows and two organ-pipes. One of the bellows and its organ-pipe are enclosed in an airtight case containing pure air; the other is supplied from the gaseous mixture. Each experiment is very short, and occupies only a few seconds. The organ-pipes sounding ut_4, and the gas mixed being marsh gas, the following results are obtained:—

For 1 per cent. of marsh gas in air there is about 1 beat per 3 seconds.

„ 2 „ „ „ 3 „ 2 „

„ 3 „ „ „ 2 „ 1 „

„ 4 „ „ „ 3 „ 1 „

And so on, the beats increasing in frequency in proportion as the gaseous mixture contains more marsh gas. When there is 12 per cent., there are about 9 beats per second; with 20 per cent. they become very rapid, and with 25 per cent., extremely rapid, but always very distinct and clear.

The Hardy fire-damp detector gives similar results in mixtures of air and carbonic acid gas. In a few hours the respiration of a single person adds sufficient carbonic acid gas to the air of a room for the gas-detector to indicate its presence.

Instruments intended for testing gaseous mixtures whose density is almost the same as pure air, should be fitted with an organ-pipe giving a higher note; ut_5, for instance.

This fire-damp detector is especially intended for the testing of the presence of fire-damp in mines, may take any form, and will afford continuous indications, not only in the mine, but also in the office of the mining engineer. M. W. B.

THE CHESNEAU FIRE-DAMP INDICATOR.

(1) *Essais effectués dans les Mines avec l'Indicateur de Grisou de G. Chesneau. By* G. CHESNEAU. *Annales des Mines*, 1893, *series* 9, *vol. iii., pages* 509-531.

(2) *Instruction pour l'emploi de l'Indicateur de Grisou de G. Chesneau.* IDEM. *Annales des Mines*, 1893, *series* 9, *vol. iii., pages* 532-544, *and figures.*

The Chesneau fire-damp indicator has already been described in the *Transactions.**

The first paper is a report submitted to the French Fire-damp Commission, and recounts the results of experiments made with fifteen of the Chesneau lamps in fiery mines of the various coal-fields of France. Check experiments were conducted at the Paris School of Mines, and in the laboratories of the St. Etienne and Ronchamp collieries companies, the total number of comparisons made amounting to 200. Generally speaking, the results proved so satisfactory that several colliery managers proceeded to use the indicator in their fiery pits, without waiting for any official decision as to their value, and the chief engineer of the Anzin collieries announced his intention of substituting the Chesneau indicator for the Pieler lamp.

The author sums up the improvements which he has made in this fire-damp indicator, and he draws special attention to the use of cotton-wadding in the spirit reservoir. When the copper chloride solution (mixed with the methylated spirit) comes into contact with the metallic copper wall of the reservoir, an insoluble sub-chloride tends to separate out, clogs the wick, and arrests the ascension of the alcohol, so that the burning wick carbonizes and the flame-caps rapidly grow dim; the wadding absorbs the sub-chloride and obviates all these inconveniences. The alcohol should be of constant specific gravity of 0·8275 at 60 degs. Fahr.

That the Chesneau lamp is extinguished immediately in explosive mixtures has been proved repeatedly, and the author states that a Government engineer took it about with him for 4 hours (in workings where no one would have dared to take a Pieler lamp) in atmospheres containing 2 to 4 per cent. of fire-damp, without its becoming noticeably heated or its regulation being disturbed. As to the clearness of the flame-caps with less than 1 per cent. of fire-damp, the Chesneau indicator has been on several occasions experimentally compared with the Pieler lamp. much

* *Trans. Fed. Inst.*, vol. iv., page 617.

to the disadvantage of the latter. The indications given by the Chesneau indicator agree, on the whole, very closely with the laboratory analyses of the pit air made to check the experiments; and it has never, even in dusty mines, indicated fire-damp when the analyses showed that none was present. All tends to prove that the lamp, in addition to being a good indicator, is equivalent to a good safety-lamp, and can be confidently used in any part of a mine. O. S. E.

THE BORINGS FOR NATURAL GAS IN THE TERTIARY STRATA OF UPPER AUSTRIA.

Neue Tiefbohrungen auf brennbare Gase im Schlier von Wels, Grieskirchen und Eferding in Oberösterreich. By G. A. Koch. *Verhandlungen der k.k. geologischen Reichsanstalt, Vienna*, 1893, *vol. v., pages* 101-129.

The Schlier formation is defined by Ritter Von Hauer as consisting of marls and sands with few organic remains; it is of Neogene age, probably late Miocene. The present paper is in continuation of a report published by the author in 1892. Within the municipal boundaries of the town of Wels no less than 16 borings had been put down by Easter, 1893, and the author gives detailed notes on each. Taking them in the order in which they are enumerated by him, we have:—

No. 1.—Depth, 820 feet. The average yield of gas is estimated at 5,300 cubic feet per 24 hours, but as a great deal is lost through defective appliances, it is believed that the real amount is nearer 7,000 cubic feet. The yield is larger with a low than with a high barometer. Although this is surrounded by new borings, the gas comes off in apparently undiminished quantity; it is used for light and fuel. Water is squirted out of the shaft by the gas-pressure.

No. 2.—Depth, 886 feet. Yield too small for industrial purposes. Brackish water abundant.

No. 3.—Depth, 558 feet. Yields a good deal more gas than the former; the two, put together, are stated by the landowner to yield between 353 and 530 cubic feet of gas per day. The gas is chiefly used for heating purposes.

No. 4.—Depth, 886 feet. Gas in moderate quantity. Water comes up by bursts; it contains, among other alkalies, magnesia and ammonia. The gas, colourless, odourless, and combustible, was analysed, with the following result by volume:—Carbon dioxide, 0·7; oxygen, 1·9; nitrogen, 16·5; methane, 79·7; and carbon monoxide, 1·2. Its heating power is somewhat greater than its illuminating power, but the latter is much improved by the admixture of a very small quantity of coal-gas. It is used for light and fuel.

No. 5.—Boring put down by the municipality in the public gardens. Depth, 677 feet. When the boring had reached 374 feet, a curious crackling sound gave warning of the uprush of gas, and gas continued to come off down to 656 feet. On account of the breakage of rods and the striking of hard strata the operation of boring has been temporarily suspended. (The strata of the Schlier become harder as one approaches the Traun river, near which this bore-hole is situated.)

No. 6.—Depth, 590½ feet. Gas-yield abundant. It is used for heating and lighting the owner's residence.

No. 7.—Boring stopped at a depth of 689 feet. Gas comes off slowly.

No. 8.—At Wels Castle a boring was carried down 984 feet. At 505 feet gas began to come off, spirting up water before it.

No. 9.—Boring stopped at a depth of 640 feet. Odourless gas came off in large quantities. It is used for heating purposes.

Nos. 10 and 11.—In the garden of the Catholic Working Men's Club. The first boring went on satisfactorily till a depth of 292 feet was reached, when the tool

struck a lump of quartz and snapped. The next boring is closely adjacent, and attains a depth of 512 feet. A doubtful estimate puts the gas-yield at 565 cubic feet daily.

No. 12.—Depth, 551 feet. Gas distributed in twelve burners by pipes about $\frac{1}{2}$ inch internal diameter, burns day and night, giving sufficient illumination for the underground (basement) portions of a cement factory.

No. 13.—Depth, 394 feet. Plenty of gas, but no water.

No. 14.—Depth, 697 feet. The gas-yield was said to be double that of No. 1 well, *i.e.*, 10,594 cubic feet per day at least.

No. 15.—Gas was first met with at 171 feet.

No. 16.—Gas met with at 196 feet, boring carried beyond 370 feet. No water. Gas partly used for firing boilers at a steam saw-mill.

So much for the results obtained at Wels, but the boring fever has taken possession of the people in other districts of Upper Austria, which lie upon the same geological formation, and the following trials are reported:—At Willing, parish of Neukirchen-Lambach, a boring went down 656 feet without striking gas-bearing strata. At Haiding gas was found at a depth of 230 feet, but the boring apparatus was so clumsily handled that operations will have to be started afresh. At Grieskirchen, north-west of Wels, the Schlier formation all but crops out at the surface, and a boring put down there in March, 1893, attained a depth of 918 feet. Combustible gases came off in considerable quantity; within the last 32 feet a bituminous stratum about 20 feet thick was pierced through. No trace of petroleum was found, although the borehole is not $\frac{1}{4}$ mile distant from a so-called petroleum spring. At Simbach and Lahöfen artesian wells have yielded water reeking with sulphuretted hydrogen.

The few analyses that have been made of waters from the Schlier point to the occurrence in some cases of bromides and iodides, together with sodium chloride, and the author suggests that the strata are of marine origin. In his general summary he points out that, taken as a whole, the gas-yield is abundant, nor does it appear to diminish with time. Bearing in mind the great thickness of the Schlier formation, and the vast horizontal extent of the gas-bearing strata in Upper Austria, as also the relatively insignificant number of borings, there is no reason to apprehend exhaustion of the natural gas-reservoirs for many years. Whenever that exhaustion does take place, the American example may well be followed by searching deeper down for other gas-bearing strata.

It will be seen that gas is already tapped at very different depths in the Schlier, from 115 down to 820 feet and beyond. Within the Wels municipal boundary three distinct gas-bearing zones may be recognized: (1) a southern zone, parallel with the Traun river, poor in gas; (2) a zone parallel to the first, not so poor; (3) the Town Moor zone, much the richest in gas of the three. It is considered extremely probable by Dr. Koch that petroleum or ozokerite will ultimately be discovered in paying quantities in these districts. O. S. E.

ANCIENT GOLD-MINES IN BOSNIA.

Ueber Goldgewinnungs-stätten der Alten in Bosnien. By H. B. von Foullon. *Jahrbuch der k.k. geologischen Reichsanstalt, Vienna*, 1892, *vol. xlii., pages* 1-52, *with map, plate I., and five figures.*

A short summary of the bibliography of the subject is followed by a geological description of the mountainous district which stretches east of the upper course of the Vrbas river. Generally speaking, the rocks consist of Palæozoic slates and

limestones, unconformably overlain by Triassic beds (Werfen shales), of unfossiliferous fresh water Tertiary strata, masses of eruptive rock, and great spreads of drift and alluvial deposits. The two last-named are of paramount importance in connexion with the occurrence of gold.

The drift varies in character, but is usually a very coarse gravel, largely made up of pebbles of eruptive rock, ironstone, and manganese-nodules. The alluvial deposits, of course, comprise representatives of all the rocks in the district, but, in contradistinction to the drift, the chief components are limestone and slate. It is noted that quartz is of rare occurrence either in the drift or the alluvium, and when it is found, its origin can be traced to certain lenticular masses in the slates.

The eruptive rocks have often a distinctly stratified appearance, the stratification being conformable with that of the Palæozoic slates over which they are spread; and the evidence points to the vicinity of the headwaters of the Suhodol burn as the probable centre of eruption. These rocks are described as quartz-porphyries, more recent than the Palæozoic slates, but older than the limestones.

A detailed account is then given of the abandoned diggings and old waste-heaps, which still bear witness to the mining activity of the ancients, the following topographical subdivision being adopted:—

1.—The diggings in the upper valley of Vrbas.
2.—The diggings in the higher mountain-region.
3.—The old workings in the Rosinj district.
} Vrbas river-basin (upper portion).
4.—The diggings in the Lasva valley.
5.—The diggings in the Fojnica-Zeleznica valley.
6.—The diggings in the lower mountain region along the Fojnica-Zeleznica.

Local traditions import that gold-mining went on here, more or less interruptedly for over two thousand years, from the days of the Romans to the beginning of the present century. The industry appears, indeed, to have been pre-Roman, the imperial conquerors merely continuing and expanding the work begun by the aborigines. However that may be, the ancients, invariably proceeding in the uphill direction, appear to have systematically washed round about Gornji Vakuf every bit of drift or alluvium that would repay the trouble. They were quite aware of the commercial value of the pockets of drift occurring in clefts, crevices, hollows, or basins; and the testing of samples from the drift-material, wherever it has been left untouched by the ancients, shows it to be extremely poor in gold. Such gold as it does contain is very finely divided, dispersed in the form of minute, excessively thin flakes.

At the old diggings of the Zlatno Gueno (or golden threshing-floor), abutted a water-conduit six miles long, carried round steep mountain-ridges, and a double conduit of the same kind is traced up to the Cervena Zemlja (or loam-heap): the antiquity of this enormous waste-heap may be gauged from the gnarled timber which now clothes its lower slopes. The author infers that labour was cheap in those bygone days, rather than that gold was extraordinarily abundant in the deposits of the Vrbas river-basin. There is no trace of metalliferous veins here, nor of any mining operations such as would have been conducted to work veins.

As to the origin of the gold in the Vrbas river-basin, with a view to the determination of that point, thirty samples from as many different localities were washed and analysed. They all contained red hæmatite, magnetite, titanite, zircon; several contained brown hæmatite, manganese ore, baryta, rutile, tourmaline, etc., and four samples contained cinnabar: only five are mentioned as not containing free gold, in a very finely-divided state. Now, nearly every one of the constituents of these samples are also found in the sand which filled up a limestone-cavern at the Bosanska-Idria mine on the Zec. The limestones which bulk largely in the mountain *massifs* contain quicksilver and fahlores, and the author's theory appears to be that

the gold, in drift and alluvium alike, is transported material resulting from the decomposition and degradation of those rocks, as also from the denudation of the clay-slates, but not from the quartz-porphyries. The ancients evidently expected to find, and possibly did find, some gold in the slates; for they dug pretty deeply into the solid rock in a good many places; and the fact that no old waste-heaps are visible near the depressions thus left, merely shows that the material was taken away to some convenient spot, and stamped and washed there.

The Lasva valley was never so active a gold-washing centre as the Gornj Vakuf (Vrbas river) district; while, concerning the Fojnica-Zeleznica valley, tradition curiously enough dwells on rich silver-mines, for which there is not a shred of evidence, but there are unmistakable traces of mercury and gold-mining having been carried on. O. S. E.

THE GOLD-DEPOSITS OF THE PUNA DE JUJUY, ARGENTINE REPUBLIC.

Los yacimientos auriferos de la Puna de Jujuy. *By* V. Novarese. *Anales de la Sociedad Cientifica Argentina, Buenos Aires,* 1893, *vol. xxxv., pages* 89-117.

The Puna de Jujuy is a region of high, treeless, arid table-land, bounded by mountain-ranges running north and south, parallel with the general trend of the main chain of the Andes. The most ancient rocks of the district are schists and grauwackes, several thousands of feet thick, which contain, the latter especially, pyrites-crystals in great abundance and gold-bearing veins. In this formation no fossils have so far been discovered, but Dr. Brackebusch, reasoning from analogy with similar rocks in neighbouring districts, holds that it is of Silurian age. The schists and grauwackes are succeeded by unfossiliferous, vividly-coloured, red and yellow sands, more or less marly shales, dolomitic limestones of varying structure, and another series of sands, distinguishable from afar by their strikingly varied colouration. These have been grouped together as Lower Cretaceous.

The two groups of strata above described are folded and distorted on a magnificent scale; above them lie almost horizontally Tertiary sands, clays, and micro-conglomerates. These are followed by a coarse conglomerate of Quaternary age, which is gold-bearing; it is made up of fragments of white, grey, and green Silurian quartzite, cemented together by a grey or red argillaceous matrix. This conglomerate runs in great terraces along the mountain-valleys, and forms the fine plateau on which is built the capital of the province, the city of Jujuy. Moreover, gold-bearing alluvia of later age occur, and efflorescences of sodium chloride and various sulphates on the surface-soil are common. In the dry season, when the mountain torrents disappear, their beds are found covered with a saline crust. Ancient lava-flows and porphyritic and trachytic cones are numerous; they are certainly of post-Palæozoic, and probably of Tertiary, or even post-Tertiary age.

The gold-deposits are almost entirely limited to the western portion of the Puna (departments of La Rinconada, Santa Catalina, and part of Cochinoca). Those of the Sierra de Cochinoca await more thorough exploration. The most important known deposits occur in the range which divides the Rio Grande de San Juan de Mayo from the high table-land, the Sierra de Cabalonga or De La Rinconada: this Sierra runs north and south for about 65 miles, and varies in width from 6 to 8 miles.

Auriferous white quartz-veins occur in the schist-grauwacke formation, preferably in the schists. They are numerous, and follow the general strike and dip of

the country rock. The quartz is crypto-crystalline, compact, and as a rule extremely hard; crystals of gold the author failed to see; the native metal invariably appeared in small amorphous agglomerations. One peculiarity is that the gold at times impregnates the country-rock, so that the latter is often richer than the quartz itself. Various more or less desultory attempts have been made to work the veins industrially, but all have been successively abandoned. The author collected some specimens which were assayed at the mint at Buenos Aires, and yielded the following results :—

		Auriferous Quartz.	Country Schist.	Country Schist excavated and found in the Old Workings.
		Grammes.	Grammes.	Grammes.
La Perdida mine, Sta. Catalina ...	Gold per ton	15	112·5	64·94
La Cruz del Sud mine, Timon Cruz, headwaters of the Rio Sta. Catalina	,,	179·9	5·0	—

From the latter also, a galena, with much quartz-gangue, yielded per ton 217·3 kilogrammes of lead, 235 grammes of silver, and 5 grammes of gold. The small quantity of material collected by the author did not permit of analysis being made of the gold itself, as found in the veins.

The author, following the classification adopted by professional miners, subdivides the gold placer, etc., deposits as follows :—

1. Superficial deposits (*aventaderos* or shallow placers), evidently of Quaternary age; these include (*a*) placers *in loco*, formed by the degradation in place of the vein-material; (*b*) the bottom of the actual valleys and the beds of modern rivers; (*c*) the now abandoned beds of actual rivers in the drift-terraces of their valleys.

2. Deep-seated deposits (*veneros* or deep leads) covered by more or less considerable masses of barren material distinct from the auriferous stratum.

The gold-bearing conglomerate, composed chiefly, as above stated, of pebbles of Silurian grauwacke and schist, is further characterized by cubical crystals of pyrites, locally termed *binches*, the abundance of which is regarded by the miners as an indicator of the richness of the conglomerate (this, however, is not invariably a safe guide), and by the presence of an extremely fine, magnetic black sand which is always found in the washings to accompany the gold. There is an idea prevalent in the district that rubies occur in the auriferous conglomerate, but these turn out to be fragments of rose quartz; the author, has, however, seen crystalline fragments, which appear referable to topaz and garnet. The richest mine in La Puna is that belonging to category *a*, of La Perdida, already mentioned in connexion with the quartz-vein deposits; despite the very imperfect method of washing which there obtains, the daily production in gold nuggets averages 30 grammes per cubic metre of conglomerate—this, without counting the gold-dust, which practically all goes to waste.

In category *b* the gully or Quebrada de Ajedrez and the Quebrada de Llaucana are good examples. Here the conglomerate-gravel is richest near its base; the nuggets show many traces of the action of water, being rounded and sometimes flattened.

In category *c*, the Buena Vista concession, north of the Cerro Galan peak, along the southern course of the Rio Grande de San Juan de Mayo, may be cited. The deposits here yield very pure gold in nuggets and flakes, averaging 3 grammes of gold per cubic metre of material.

Of the deep leads the Eureka mine (14,181 feet above sea-level) at Tagarete, is the sole remaining example in La Puna. The conglomerate here is of great thickness,

and its components attain enormous dimensions, but the auriferous seam is much less coarse, and the cementing-matrix is softer than in the mass of the rock. It is characterized by a roof containing agglomerations of native copper and copper ore, cemented by a vividly-coloured green and blue matrix. This band is as rich in gold as the non-cupriferous band. The average gold production of the Eureka mine is 5 to 6 grammes of rich-coloured nugget gold per cubic metre of conglomerate; the process of washing is very imperfect. A small nugget from this mine was found on analysis to contain gold 95·7 per cent., silver 3·86 per cent., and iron 0·427 per cent. O. S. E.

GEOLOGY OF THE TUNISIAN ATLAS.

Beiträge zur Kenntniss des tunisischen Atlas. By A. Baltzer. *Neues Jahrbuch für Mineralogie, Geologie, und Palæontologie, Stuttgart*, 1893, *vol. ii., pages* 26-41, *and plate III.*

A short description is given of the topography of the mountain-region as distinct from the plain-country of Tunisia, the author dividing the former area as follows:—

1.—Northern zone, forming the continuation of the lesser Atlas, bounded to the north by the Mediterranean and south by the Medjerda river. Here are the Khroumir Hills with fine cork-oak forests, and the Nefzas Hills with metalliferous rocks.

2.— Middle zone, forming the continuation of the great Atlas, with a well-defined Jurassic chain. All the mountain-ranges in this zone run north-east, and are deflected to the north as they approach the plain of Kairouan.

3.—Southern zone, bounded south and west by the depression of the Chotts (salt lakes) and east by the Sahel (a fertile plain).

The entire mountain-region of Tunisia is an area of plication, a folding movement which lasted into Pliocene times; and along a north-and-south line about 200 miles in extent, which may be drawn from Tabarka to the Chotts, the author reckons no less than twenty-five hill-ranges or great folds.

He describes in some detail the Zaghouan chain. Here in the grey, variously coarse and fine-grained, crystalline, massively-bedded limestones of Jurassic age, which form the great mass of the mountains, occur deposits of zinc ore (carbonates and silicates). These ores are associated with galena, blende, baryte, and orthoclase, they have only been worked within the last few years, and the author noted about a dozen *attaques* or diggings, mostly very shallow. The zinc ore is roasted, shipped at Tunis to Antwerp, and forwarded thence for further treatment to La Vieille Montagne. Jurassic limestone with zinc ores and galena also occurs in the Jebel Resas, north east of the Zaghouan chain, and at the Bou Kournine, near Hammam Lif. The zinc ores occur variously as stratified deposits, infillings of cracks and fissures, concretions, and impregnations. The sections in the plate show the localities where the ores are found, but otherwise this abstract contains almost every particle of information concerning the ores that can be gleaned from the original paper. Mr. Baltzer appears to be more concerned with the tectonic geology, and the comparison of the Tunisian Jurassic with the Swiss, than with matters which more immediately interest the mining engineer. O. S. E.

GEOLOGY OF THE TERRITORY OF NEUQUEN, ARGENTINE REPUBLIC.

Neuquen. *By* FRANCIS ALBERT. *Boletin del Instituto Geográfico Argentino, Buenos Ayres*, 1893, *vol. xiv., pages* 154-176.

Coal-deposits are said to extend over large areas, and where trials have been made the deposits seemed to increase in depth. Two samples analysed, one in Paris and the other at Buenos Ayres, were reported of good quality, ash 2 per cent. Four petroleum-wells are known, and the oil is used by the inhabitants for illuminating purposes. Finds of copper, iron, and silver ores are known, as well as argentiferous galena of so high a grade that a Chilian company found it pay to have the ore carried on mule-back over the mountains into Chili for treatment. Other minerals known to occur abundantly are sodium nitrate, rock-salt, native sulphur, amber, jet, marbles and plastic clays of various kinds. Alluvial gold, yielding sometimes fairly large nuggets, has been found in the sands of at least one stream, and gold-washing is now going on.

Two leagues west of the pastoral park-like district of Los Pinos rises the (moribund) volcano, Mount Copahues, with its sixty or more thermal springs of different chemical composition and varying temperatures. Other thermal springs are known at El Domuyo and Chapún ; these have formed great deposits of native sulphur.

The projected Interoceanic Railway from Buenos Ayres harbour to Talcahuano harbour in Chile, would pass through the territories of Pampa Central and Neuquen, crossing the Andes at Antuco. It is stated that the French engineers who made a reconnaissance-survey of this route in 1889-1890, considered that the greatest piece of work, the tunnel under the Andes, need not be more than ½ mile long, a statement which certainly appears to require confirmation. O. S. E.

THE GEOLOGY OF NORTHERN PATAGONIA, ARGENTINE REPUBLIC.

Zur Geologie von Nord-Patagonien. *By* JOSEF V. SIEMIRADZKI. *Neues Jahrbuch für Mineralogie, Geologie, und Palæontologie, Stuttgart*, 1893, *vol. i., pages* 22-32.

The author started from Buenos Ayres in November, 1891, in charge of a small exploring party, to examine the comparatively little-known districts inhabited by the Ranqueles and Manzaneros Indians, districts now termed officially Pampa Central, Rio Negro, and Neuquen.

Drawing a section from the mouth of Rio de la Plata west-south-west across the Pampa Central, the first formation met with is the Pampa proper, consisting of the enormous alluvial deposits of the great river Parana. Below these are Pliocene shell-beds. The author regards the Pampas formation, characterized by its gigantic fossil *Glyptodontia*, as equivalent to the older drift of Europe. Here and there outliers of Miocene and Eocene rocks rise like islands above the Pliocene plain, as, for instance, Parana and Cerro Hermoso, near Bahia Blanca. Taking a line from the last-named locality north-west to San Luis, we cross a country abounding in lakes, which have, however, no outflow, and we ascend a gradually rising declivity up to 300 metres (984 feet) above sea-level, leading us on to the flat, high, table-land of the Pampa Central. Though all the lakes, even the so-called freshwater ones, are more or less salt, no trace of marine organisms could be found. The uppermost formation in the Pampa Central is a white, calcareous marl (*toscu*), which gradually gives place to a conglomerate consisting of rounded porphyry-pebbles, lying unconformably on every other rock. Silicified tree-stems are found in the greyish-white marly sandstone (probably Oligocene) which forms the crags along the Rio Negro ; this sandstone thins away between the Roca settlement and the confluence of the

Limay and the Neuquen, while the underlying dark red sandstone takes its place along the entire course of the Limay. Meanwhile the gradually rising Pampa table-land attains the level of 1,000 metres (3,280 feet) in the neighbourhood of the Cordillera. The saline nature of the soil becomes everywhere apparent, and the water of such rain-pools as are formed has, in consequence, a bitter salt taste. Gypsum does not occur.

The greatest depression in the Pampa is in the north, in the Vermejo and the Pilcomayo districts, and not (as Mr. Doering had asserted) in the great salt-lake of Urre-Lafquen, which lies 220 metres (723 feet) above sea-level. The author describes this lake as bounded south, west, and north-west by the Lihue-Calel mountains, and stretching away in other directions to an apparently limitless horizon, for there the lake is succeeded by a vast expanse of *salitrales* or salt-steppes, glittering in the sun like water.

Mountain-ranges occur dispersedly all over the region described by the author; in the north-east they consist mainly of granites and Palæozoic rocks, in the south-west of folded gneisses, granites, granitic gneisses and porphyries (Sierra Lihue-Calel), and amphibolites. In the Sierra Baya, at the German settlement of Hinojo, marble quarries are worked in the Silurian rocks. The yellow marble is topped by a quartzite of enormous thickness (this quartzite is the main component of the Tandil and Ventana mountains), and that again by a black, bituminous limestone. Three geographical miles south of Lihue-Calel the explorers came upon a ridge of mica-porphyrite (1,312 feet) containing very rich copper-ore veins. The Choique-Mahuida range which runs from the southern shore of Lake Urre-Lafquen along the right bank of the Curá-có is also porphyritic, and small porphyry ridges are seen along the route from the Rio Colorado to Choele-Choel. Great spreads of tuffs and porous lavas are traced for miles around the headwaters of the Alumine and the Collon-Cura, and along the Alumine valley rises a great chain of extinct volcanoes (6,560 feet) hitherto unmapped.

Evidence of glacial action occurs in the neighbourhood of Tandil in the shape of magnificent moraines, roches-moutonnées, ice-groovings, etc., and at the foot of the Cordillera in the shape of glacier-formed terraces. Peat-moors abound in the Cordillera, and also dangerous bogs (*menucos*) usually hidden by a luxuriant growth of herbage. These bogs are especially numerous in the vicinity of Lonquimay, on the upper Bio-bio and in Araucania O. S. E.

IRON-ORE IN NORTHERN SWEDEN.

(1) *Om Routivare järnmalmsfält i Norrbottens län.* *By* WALFR. PETERSSON. *Geologiska Föreningens Förhandlingar, Stockholm*, 1893, *vol. xv.*, *pages* 45-54, *and map.*

(2) *En ny jernmalmstyp representerad af Routivare Malmberg.* *By* HJ. SJÖGREN. *Ibid.*, *pages* 55-63.

(3) *Ytterligare om Routivare jernmalm.* *By* HJ. SJÖGREN. *Ibid.*, *pages* 140-143.

The iron-ore of Routivare, in Norrbotten, Sweden, is found in a hill rising from 520 to 585 feet above the surrounding country, and a few miles from Quickjokks. The ore consists chiefly of titaniferous magnetite, green spinel being an essential constituent, while olivine and decomposition-products of the same are subordinate. According to a series of analyses, the percentage of iron varies from 47·91 to 52·16; that of titanium from 1·35 to 13·05; and of phosphorus from 0·067 to 0·002. It contains no sulphur. The percentage of iron occasionally rises to 59. The area occupied by this ore is not extensive, being estimated at between seven and eight acres.

An eruptive rock, an olivine-gabbro, forms the hill in which the iron occurs. It is of interest to note that the ore consists chiefly of the same minerals as those that occur in the rock, since this lends countenance to the view that the former has been formed by some process of differentiation and concentration in the latter. The ore is considered to form a new type of iron-ore, and described as a magnetite-spinellite. Its commercial value is doubtful. Experimental smelting carried out in the laboratories of the Filipstad school of mines showed that the largest yield of iron (53·2 per cent.) resulted from the use of 10 per cent. each of limestone and of quartz. G. W. B.

HUNGARIAN IRON INDUSTRY.

Ungarns Eisenindustrie. Von FERDINAND BLEICHSTEINER. *Berg- und Hüttenmännisches Jahrbuch, Vienna*, 1893, *vol. xli., pages* 203-234.

The richest and purest ore and the thickest coal-seams occur in Transylvania. Yet no important works have been started there during the last ten years, and the mineral wealth of Transylvania seems to be reserved for future development.

The brown coal of Zsilthal in the department of Hunyad, is the most important deposit in Hungary. It occupies an area of 18½ by 3 miles. There are twenty seams of which eight are workable, and of which the thickness sometimes amounts to as much as 98 feet.

The coals of Borsoder and of Gömörer will play an important part in the future of the Hungarian iron industry. The seams in this case are not thick, but of great extent. They occur in strata which are much folded, and considerably faulted.

From the Gyalaer iron-ore mine, an ore, rich in manganese, free from phosphorus and sulphur, and containing 50 per cent. of iron, is obtained.

The ore used in the Borsoder works is exposed for a length of 4½ miles, and has a known thickness of 131 feet, but is supposed to average much more. An ore supplied from this mine contains over 50 per cent. of iron, occasionally rendered impure by copper pyrites. The winning of the ore is exceedingly cheap, as it is free from intercalations of barren and poor ore. It is roasted on the spot in furnaces, 30 feet high, with an expenditure of brown coal amounting to about 5 per cent. of the ore roasted. A machine is used for removing the 33 feet of marl and conglomerate which cover the ore-deposit without manual labour.

The largest mines and smelting-works in Hungary are concentrated in the Sajó-Rimathal district; the most extensive mine is that of Vashegy. It has three beds of ore, 39, 65, and 98 feet thick respectively. The lowest contains 20 per cent. of manganese and 40 per cent. of iron; the middle 2 per cent. of manganese and 50 per cent. of iron; and the uppermost often up to 60 per cent. of iron. Unfortunately the ore is not free from sulphur and copper, is difficultly fusible, and requires much lime in the furnace.

North Hungary possesses an area of many square miles of iron-ore, with an incalculable extent of forests for fuel. In other parts, as in the Banate, the iron-ores are poorer, and iron production more costly.

In Croatia, although iron-ore occurs not infrequently, the iron industry—partly on account of the cost of transport—has attained no marked development.

Smelting of iron-ores was carried on in Hungary in favourable places in early times. From this early beginning the present iron-smelting industry was developed at the end of last century. In bygone days charcoal could be obtained abundantly and cheaply from the extensive forests, but now it is dearer because the wood is

used for other purposes. In North Hungary small charcoal-furnaces, often worked with blowing engines and heated air, are in use. They produce grey iron. In 1891 there were 64 furnaces at work in Hungary, and 9 idle. G. W. B.

FRANCKEITE, A BOLIVIAN SILVER-ORE.

Ueber Franckeit, ein neues Erz aus Bolivia. By ALFRED W. STELZNER. *Neues Jahrbuch für Mineralogie, Geologie, und Palæontologie, Stuttgart*, 1893, *vol. ii., pages* 114-124.

The mineral herein described is found in the district of Animas, a short distance south-east of Chocaya, and about 13,060 feet above sea-level. Chocaya is a township of the province of Chichas, Department of Potosi, and is situated about 105 or 110 miles south-south-west of the well-known city of that name. In Animas six argentiferous veins are known, bearing iron and copper pyrites, zinc blende, galena rich in silver, *Ilicteria* (franckeite), *polvorilla* (silver sulphide), argentiferous fahlores, dark ruby silver ore, and native silver. Here and there Jamesonite and Wurtzite also occur. The gangue-materials are quartz and kaolin, and less frequently baryta.

At Chocaya, clay-slates, presumably of Palæozoic age, predominate, and these are traversed by igneous rocks (dacites); the metalliferous veins are of later formation than the dacites.

The ore *Ilicteria*, described as consisting of lead, zinc, tin, and silver compounds, is invariably accompanied by *polvorilla*. In the Veta del Cuadro mine, *Ilicteria* occurs in direct association with iron pyrites and zinc blende as the latest band of the three forming the *veta* or vein. The mineral has the form of very thin flakes arranged radially like the petals of a flower. *Ilicteria* from San Juan is in little spherules of radiate structure, average diameter $\frac{1}{5}$ inch, ranging up to about $\frac{1}{3}$ inch, sometimes massed together in reniform agglomerations. The mineral cleaves very perfectly in one direction, it is fairly malleable, makes slight marks on paper, has a greasy feel, and as it apparently does not scratch calcite its hardness is estimated at 2·75. The specific gravity is 5·55. The colour is blackish-grey to black, and the mineral possesses metallic lustre; it is opaque, even in extremely thin flakes examined under the microscope.

The chemical analysis was undertaken by Dr. C. Winkler, with the following results:—Heated in a tube closed at one end, from which the air has been completely expelled and replaced by carbon dioxide, a sublimate of germanium sulphide (GeS) comes off, rapidly giving place, on the admisson of air, to a thin film of germanium oxide (GeO_2). Heated in hydrochloric acid the powdered mineral gives off hydrogen sulphide, but dissolves only very sparingly. In *aqua regia*, on the other hand, it dissolves rapidly and easily, sulphur separating out. It also dissolves in hot nitric acid, leaving then a residue of white powder, consisting of the oxides of antimony, tin, and germanium. The exact quantitative determination of germanium was impracticable, on account of the small quantity of material available; the analyst estimates the percentage of germanium at 0·1, and there is also a fractional percentage of silver. Pure *Ilicteria* is estimated to contain:—Lead, 55·55 per cent.; tin, 13·56 per cent.; antimony, 11·55 per cent.; sulphur, 19·34 per cent. This analysis leads to the formula $5PbS + 2SnS_2 + Sb_2S_3$. Dr. Kollbeck subsequently determined the silver content of several specimens in the dry way, and found the varying percentages of 0·857, 1·040, and 1·037 Ag. The variation is probably due to the very intimate association of the *Ilicteria* with *polvorilla* and other ores richer in silver, and it is very possible that in hand-picking in large

quantities the miners cannot very well separate one from the other; hence, the perhaps undeserved reputation (in Bolivia) of *Ilicteria* as a silver-ore. In its form, in its physical properties, and in its qualitative composition this mineral strikingly resembles plumbostannite (already known from Moho in Peru), but not in quantitative composition. The difference will be seen by comparing with the above analysis of *Ilicteria* that of plumbostannite given by Dr. Raimondi:—Lead, 30·66 per cent.; tin, 16·30 per cent.: antimony, 16·98 per cent.: sulphur, 25·14 per cent.; iron, 10·18 per cent.; zinc, 0·74 per cent. Under all the circumstances, the author concludes that *Ilicteria* is a new mineral species, for which, in compliment to the mining engineers, Messrs. Carl and Ernst Francke, he proposes the name Franckeite.

O. S. E.

THE NEW BOLIVIAN SILVER MINERAL CYLINDRITE.

Ueber den Kylindrit. By A. Frenzel. *Neues Jahrbuch für Mineralogie, Geologie, und Palæontologie, Stuttgart*, 1893, *vol. ii., pages* 125-128.

The name cylindrite is applied to this mineral because of its remarkable cylindrical or rolled form; in section, the cylindrical rolls show a concentric structure reminding one of the appearance of a roll of paper, and looked at singly, they appear to the observer more like a piece of man's handiwork than a natural product. The mineral occurs at the Santa Cruz mine, Poopó (Bolivia), a district where for many generations silver, tin, copper, and lead-ores have been worked in the Silurian strata of the Cordillera (12,431 feet above sea-level).

Cylindrite has a bright metallic lustre, blackish leaden-grey colour, a black streak, and makes black marks on paper. It is moderately brittle, its hardness varies from 2·5 to 3, and its specific gravity (according to two determinations) is 5·42. Chemical analysis gave the following percentage results:—Lead, 35·41 per cent.; silver, 0·62 per cent.; iron, 3·00 per cent.; antimony, 8·73 per cent.; tin, 26·37 per cent.; sulphur, 24·50 per cent.; total, 98·63 per cent. The quantitative proportions, deduced from this analysis are:—3·1 Pb Ag Fe; 1 Sb; 3 Sn; 10·5 S; suggesting the formula:—3 PbS. Sb_2S_3 + 3 (PbS. 2 SnS_2).

Germanium was specially tested for, but not found. Acids in the cold do not attack the mineral, but hot hydrochloric acid dissolves it gradually, and so too hot nitric acid, sulphur and the insoluble white oxides of tin and antimony separating out. It may be noted that cylindrite, plumbostannite, and Franckeite occur along a line ranging west of Lake Titicaca y Aullagas, from Huancané in Peru to Potosi in Bolivia. The first-named mineral is the southernmost occurrence, the second the northernmost, and the third lies between the two.

O. S. E.

MANGANESE ORES.

Minerais de Manganèse analysés au Bureau d'Essais de l'Ecole des Mines de 1845 *à* 1893. *By* Adolphe Carnot. *Annales des Mines*, 1893, *series* 9, *vol. iv.*, *pages* 189-212.

Ores of manganese are principally employed in the metallurgy of iron, in the preparation of oxygen, chlorine, and in glass works. This paper records the analyses made between 1845 and 1893 in the laboratory of the School of Mines, Paris, of 207 manganiferous ores, of which 145 are of French origin, and 62 from foreign countries, viz.:—Germany, 7; Spain, 28; Greece, 2; Austria, 1; Italy, 5; Russia, 6; Switzerland, 3; Turkey in Europe, 3; and Turkey in Asia, 7.

In the analyses care has been taken to determine the amount of phosphoric acid, of silica and clay (alumina?), of iron, copper, and cobalt oxides, and of the alkaline earths (baryta, lime, magnesia); and in every case the proportion of metallic manganese has been calculated. In fact, the tables of analyses have been so arranged as to be specially useful to glass manufacturers, iron founders, and the owners of chemical works. M. W. B. AND O. S. E.

MERCURY MINING IN THE WIPPACH VALLEY, CARNIOLA, AUSTRIA.

Bericht über den Stand des Quecksilber-bergbaues im Wippachthale in Innerkrain. By CARL MOSER. *Verhandlungen der k.k. geologischen Reichsanstalt, Vienna*, 1893, *pages* 238 *and* 239.

The discovery of native mercury two or three years ago in the Eocene sandstones of Mance near Wippach impelled a local company to make trial-diggings for the metal in that district. A preliminary trial was made at the precise spot where the mercury had first been found, but when it was ascertained that the mercury dwindled away in depth and that nothing else in the shape of an ore-formation was visible, the company commenced washing the sands and gravels accumulated at the mouths of the Wippach springs. Here quicksilver was found in little drops amongst the alluvia, and this find encouraged the promoters to have an adit driven, 335 feet long, in the slope of Mount Nanos in the direction of the presumed subterranean course of that branch of the Wippach which gushes forth at the Vollmundloch; they came here to loamy beds full of angular limestone-fragments, but the adit was ultimately abandoned because the repeated invasion of water made continuous working impossible. A similar result attended the mining operations started in the Bela cave beneath the Starygrad, but a third adit has been driven 144 feet into the heart of Mount Nanos, through highly fossiliferous Cretaceous limestones, and is to be continued by means of the Brand boring-machine, for which water-power will have to be got.

Several experimental washings of Wippach sand have been made and a large sample got in the spring of 1893 assays 0·02 per cent. mercury at a depth of 3 feet. At this depth barren clays and loams mixed with limestone-pebbles suddenly come in; and not only is the thickness of the mercury-bearing sands insignificant, but their area is small. The author concludes that they are not worth working from the industrial point of view, and suggests the possible discovery of the original mercury deposit within Mount Nanos. O. S. E.

THE MEERSCHAUM OF PRNJAVOR, BOSNIA.

Meerschaum aus Ljubic-planina bei Prnjavor in Bosnien. By M. KISPATIC. *Verhandlungen der k.k. geologischen Reichsanstalt, Vienna*, 1893, *pages* 241-242.

Chemical analysis shows that this mineral, worked on the north-western slope of the Ljubic-planina for the manufacture of tobacco-pipes, is of very similar composition to the meerschaum occasionally found in Asia Minor and Greece. The percentages are: silica, 61·09; magnesia, 25·87; iron sesquioxide, 2·59; and water, 10·47; plus 14 per cent. of hygroscopic water from an air-dried sample. There are no traces of carbonates or of lime. Magnesite is often found to accompany meerschaum, containing as an impurity more or less hydrosilicate of magnesia.

O. S. E.

PETROLEUM IN FRANCE.

Couches à Pétrole des environs de Pechelbronn (Basse-Alsace). Températures exceptionnellement élevées qui s'y manifestent. By A. DAUBRÉE. *Comptes Rendus hebdomadaires des Séances de l'Académie des Sciences*, 1893, *vol. cxvii., pages* 265-269.

Introduction.—Within the last twelve years a complete change has occurred in the working of the petroleum contained in the Tertiary measures near Pechelbronn, in Lower Alsace. Instead of a slow and troublesome working of the bituminous sand by pits and drifts, a new, more simple, and more productive process has been introduced. Bore-holes have tapped the mineral oil, which sometimes rises to the surface, or sometimes requires to be raised to the surface by pumping.

This new mode of working has disclosed many facts worthy of interest, especially a subterranean wealth in petroleum and hydrocarbon gases, whose existence has hitherto been unsuspected, as well as abnormal subterranean temperatures.

Production.—The drowning-out of a deep drift by petroleum in 1881 suggested to Mr. Le Bel the idea of using bore-holes instead of underground mining. The first borings, although not exceeding a depth of 450 feet, found flowing wells of petroleum, and the first dozen wells produced from 40 to 50 tons of mineral per day. In consequence of this unexpected discovery, a concession was acquired by a company, who made extensive explorations, both in the neighbourhood of the existing field as well as in other parts of the area conceded, and carried out the exploration to much greater depths.

At the present time the borings are more than 500 in number, and although not very far apart, they have furnished very different results. Some have produced nothing, in others the subterranean gases have forced the mineral oil to the surface with considerable violence. Since 1882, 41 flowing wells have been discovered. The flowing wells are not usually of long duration, seldom more than $3\frac{1}{4}$ years. From one of them (No. 186), discovered in 1884, 8 tons per day are still produced. Usually the force of the gases is gradually reduced, and pumps are finally used to pump the oil, accompanied with brine.

The volume of the subterranean gases is sometimes very large. One of the wells (No. 394) produced a volume of 400,000 to 500,000 cubic feet of gases per day, together with mud and small stones, for a period of six weeks, during which time its intensity gradually decreased. The flow of petroleum is usually so sudden that the workmen are soaked with it.

The pumps worked at low-pressure wells usually furnish brine; at one of them, near Kutzenhausen, there was 19·7 per cent. of salt. Mr. Berthier, who analysed sixty years ago the mother-liquor of the Soultz-sous-Forêts salt works, which is derived from the same petroleum-bearing measures, stated that it contained no sulphates, but on the other hand, was very rich in bromine, which could be extracted with profit. These two characteristics are found in the brine-well of Kreutznach, which flows from porphyry, and which had also been examined by Mr. Berthier. The same characteristics are found in the brine from recent borings, as proved by Mr. Wilm's analyses. These two distinctive characteristics of the brine-waters of petroleum bearing strata are very remarkable.

A boring made in 1890, near Surbourg, to a depth of 810 feet, at first had a daily yield of $7\frac{1}{2}$ tons of petroleum, which gradually fell to 4 tons, the present figure. Another boring, however, more distant from Surbourg, yielded from 5 to 10 tons of petroleum per day, and has yielded 3,000 tons in one year.

The constancy of the petroleum is shown by No. 146 well, which has yielded in ten years, up to the present time, 10,420 tons of petroleum, of which 3,002·9 tons were flowing and 7,417·1 tons were obtained by pumping, which is still continuing.

The present working wells are producing 80 tons per day from both flowing and pumping wells.

It may be remarked that the produce of the principal wells would have been much greater if the diameter of the wells had been as large as those bored at Baku: moreover, the wells are not pumped at nights, owing to the oil-works being too small. The production of several wells near Pechelbronn is large, even when compared with similar wells in Pennsylvania or Baku. This abundance is very remarkable when it is compared with the previous area of working with pits and drifts. Formerly, forty workmen mined the black bituminous sand, which, treated with boiling water, yielded 1·6 per cent. of its weight of mineral oil, only producing from 70 to 80 tons of petroleum per annum.

The weight of petroleum sold since 1881, the time of the first well, until April, 1893, from the Pechelbronn royalties is 69,529·685 tons, of which 27,086·8 tons were produced prior to December 31st, 1888, and 42,442·885 tons since the re-construction of the company. These figures are equivalent to a mean annual produce of 5,700 tons; that is, seventy times greater than that produced from pits and drifts.

Temperatures.—More than forty years ago, in describing the petroleum-bearing sands in the Tertiary measures of Pechelbronn, the author stated that the workings were then shallow, and that the internal temperatures presented a more than usual increase. One water-spring, in a shaft at a depth of 230 feet, had a temperature of 57 degs. Fahr., which corresponds (the temperature at the surface being at 50 degs. Fahr.,) to a mean of 1 deg. Fahr., per 33 feet.

Thermometrical measures made in several of the wells recently bored, have not only confirmed this statement, but have shown that this increase is much more rapid than the author formerly supposed. In No. 445 well, near Soultz-sous-Forêts, at a depth of 584 feet, the water had a temperature of 75 degs. Fahr., or a mean increase of 1 deg. Fahr., per 23 feet. A well bored in Haguenau forest by a German company showed, at a depth of 2,010 feet, a temperature of 141 degs. Fahr. The temperatures taken in this boring at different depths are worthy of interest, and are shown in the following table:—

Depths in Feet.	Temperatures in Degs. Fahr.
1,000	117·5
1,083	126·5
1,181	128·7
1,312	135·5
1,378	137·7
1,575	137·7
1,673	140·0
1,772	139·0
1,903	139·0
1,969	141·0
2,034	141·0

These results are in direct contradiction to the rule derived from a very large number of observations, and especially of those made by Mr. d'Arago at the Grenelle artesian wells; the temperature in the petroleum wells augmenting rapidly in depth, the geothermometric degree diminishes as the depth is increased.*

* The geothermometric degree is the vertical height descended for an increase of 1 deg. Fahr.

Near Kutzenhausen, No. 457 well has shown a more rapid increase; as at a depth of 460 feet the temperature of the water was 86 degs. Fahr., this gives 1 deg. Fahr. per 12 feet. M.W.B.

PETROLEUM IN THE SHERANI HILLS DISTRICT, INDIA.

Geology of the Sherani Hills. *By* Tom D. La Touche. *Records of the Geological Survey of India*, 1893, *vol. xxvi., pages* 77-96, *with map and five plates.*

The only mineral of practical importance found in the Sherani Hills district is petroleum, which occurs in small quantities, near the village of Moghal Kot, where it oozes at several points from a thick band of quartzose sandstone, and is collected from shallow pits dug in the sand on banks of the river Toi.

The crude oil is of excellent quality, but it does not occur in sufficient quantities to be of commercial value. It is suggested that the flow might be increased by boring, but the geological conditions are not such as to lead one to anticipate that large accumulations of oil may be found.

The existence of coal has been reported near Moghal Kot and at Chaukhel Dhana.

M. W. B.

THE ORIGIN OF PETROLEUM.

Zur Frage über die Bildung des Erdöls. *By* J. J. Jahn. *Jahrbuch der k.k. geologischen Reichsanstalt, Vienna*, 1892, *vol. xlii., pages* 361-376.

In studying the Silurian rocks of Bohemia, the author found in a dolomite of the Pridoli valley that the hollow portions of fossils—such as the chambers of Orthoceratidæ, or the space enclosed by the two valves of a Lamellibranch—often contained a sort of nucleus of little lumps of anthracite or drops of petroleum (sometimes both substances), while the rest of the drusy hollow was taken up by calcite or dolomite-crystals. The same occurrence is noticeable in the limestones near Stolba, and not only are the anthracite-lumps and petroleum found in immediate connexion with the fossils, but they, as well as mineral wax, occur dispersedly in the mass of the rock itself. These substances, whether in intimate connexion with the marine shells or apart from them, are evidently the result of the decomposition of the animal organisms which, ages ago, were buried in the calcareous mud of the Silurian sea. The author considers that his observations herein confirm the views of Prof. C. Engler, who attributes the origin of petroleum to animal organic remains, and has succeeded in artificially producing that substance and its bye-products from animal matter. But it is now seen that the idea that the occurrence of coal in quantity necessarily implies paucity of petroleum, or the abundance of the latter the scarcity of the former, in any particular locality, is erroneous. Wherever the Silurian limestones of Bohemia contain anthracite, there too, either petroleum or bitumen is found, and one is impelled to conclude that the formation of coal by the decomposition of vegetabe matter went on concurrently with the formation of petroleum by the decomposition of animal matter. The author thinks, moreover, that some anthracite may well be of animal origin, it may arise, say, from the decomposition of masses of the chitinous skeletons of Graptolites (anthracitized Graptolite-remains actually occur in bituminous-calcareous strata

of Étage E). The conditions of temperature and pressure which Prof. Engler found necessary in his experimental manufacture of petroleum, probably obtain in nature as a result of the crumpling and crushing of the earth's crust and of the eruptions of igneous rock which not seldom accompany those phenomena. It is a striking fact in this connexion that the Stolba limestones become more dolomitized as they approach the neighbouring eruptive rock (diabase). O. S. E.

DETERMINATION OF PHOSPHORUS IN IRON, STEEL, ETC.

(1) *Sur la Détermination du Phosphore dans les Fers, les Aciers, et les Fontes.* *By* ADOLPHE CARNOT. *Annales des Mines*, 1893, *series* 9, *vol. iv.*, *pages* 5-10.

(2) *Sur la Détermination du Phosphore dans les Terres Végétales.* *By* ADOLPHE CARNOT. *Annales des Mines*, 1893, *series* 9, *vol. iv.*, *pages* 11-14.

As regards the determination of phosphorus in iron and steel, the author states that he has taken care to avoid the calcination of iron nitrate, the use of hydrochloric acid, and the transformation of magnesium phosphomolybdate into the pyrophosphate, all of which are, in his view, so many contributing causes to inaccuracy.

Five grammes of iron or steel having been dissolved in 40 cubic centimetres of pure nitric acid, and the solution then treated with 10 cubic centimetres of concentrated sulphuric acid, heated, evaporated, dried, redissolved in 50 cubic centimetres of boiling water, washed on to a filter and decanted into a half-litre flask, the silica residue which remains on the filter is freed from manganese peroxide by treatment with a little concentrated hydrochloric acid; from graphite by calcination; and the weight of white silica-residue multiplied by the coefficient 0·4667 gives the exact proportion of silicon present. The organic matter in the filtrate is got out by boiling for 30 minutes, with the addition of 1 gramme of chromic acid; then 4 grammes pure ammonium sulphate is added to the liquid, and 50 cubic centimetres of the ordinary molybdic solution at 5 per cent. is poured in, and the whole kept at 100 degs. Cent. for at least one hour. The precipitation being ascertained to be complete, the precipitate is dissolved in 30 cubic centimetres of ammonia diluted with an equal quantity of warm water; this is gradually neutralized by means of nitric acid, and the resulting yellow precipitate of ammonium phosphomolybdate is brought on to a weighed filter-paper over which the solution has been decanted, dried in an oven at 100 degs. Cent., and weighed in a wide-mouthed flask. The author claims that this second precipitation enables the analyst to obtain a residue of constant composition, free from iron and from excess of molybdic acid. The weight of contained phosphorus is found by multiplying the weight of the dried precipitate into the coefficient 0·01628. (The author made a special analysis of ammonium phosphomolybdate, and his formula differs from those of Messrs. Debry, Rammelsberg, Finkener, and Gibbs; he finds that the compound contains 1·628 per cent. of phosphorus.)

The determination of phosphorus in soils differs, of course, in its details from the above, but is like the preceding, based on the principle of double precipitation. Preliminary roasting is found necessary to get rid of the greater part of the organic substances, particularly in soils containing much humus; and the use of sulphuric acid for isolating the silica is avoided because of the possible precipitation of sparely soluble calcium sulphate. O. S. E.

NEWFOUNDLAND.

Quantity and Value of Minerals Exported from Newfoundland to the end of 1891.

Taken from the Customs Returns, as published in the Journals of the House of Assembly. By J. P. Howley, 1892.

Years.	Copper.						Nickel Ore.		Lead Ores.		Iron Pyrites.		Other Minerals and Ores.		Total Values of Ores, etc., exported.
	Ore.	Regulus.	Ingots.	Value of Ore.	Value of Regulus and Ingots.	Total Value Copper Ores, etc.		Value.		Value.		Value.		Value.	
	Tons.	Tons.	Tons.	$	$	$	Tons.	$	Tons. Cwt.	$	Tons.	$	Tons.	$	$
1854 to 1864	627½	..	..	22,980	..	22,980	..	..	2,250 0?	90,000?	..	..	..	..	112,980
1865	236	..	..	8,496	..	8,496	..	..	313 5	15,036	..	..	Slate 31,000	1,240	24,772
1866	283½	..	..	10,206	..	10,206	..	..	5 0	240	..	..	..	..	10,446
1867	79	..	..	2,370	..	2,370	..	..	..	..	..	..	50	600	2,970
1868	None.	..	..	..	..	..	..	..	0 4	8	..	..	101,000	2,020	2,028
1869	3,422	..	..	102,504	..	102,504	½	600	0 2	4	..	..	..	..	110,104
1870	5,226	..	..	167,232	..	167,232	38	9,120	..	..	..	..	..	..	176,352
1871	1,407	..	..	45,024	..	45,024	7	700	..	..	..	..	..	..	45,720
1872	4,955	..	..	588,560	..	588,560	8	2,560	..	..	..	..	..	..	591,120
1873	5,553	..	..	194,355	..	194,355	120?	3,600	..	..	..	..	..	..	197,955
1874	5,052	..	..	121,248	..	121,248	98	4,704	130 0	6,240	..	..	..	..	132,312
1875	10,618	..	..	370,666	..	370,666	17½	5,550	95 0	4,560	..	..	6,000	120	378,866
1876	25,134	..	..	614,700	..	614,700	28	2,800	38 0	1,824	..	..	6,000	120	619,324
1877	47,454	..	..	1,264,004	..	1,264,004	..	..	10 0	1,392	..	..	..	..	1,265,396
1878	35,823	750	..	788,106	34,500	822,606	..	..	2 0	100	..	..	..	..	822,706
1879	28,405	1,112½	..	511,290	44,500	555,790	..	..	..	..	..	..	..	..	555,790
1880	22,042	21	..	440,840	840	441,680	..	..	..	..	..	..	..	..	441,680
1881	27,351	..	..	547,020	..	547,020	..	..	..	..	..	..	..	..	547,020
1882	19,004	260	..	456,096	12,480	468,576	..	..	..	..	..	..	..	..	468,576
1883	11,989	353	..	239,780	16,944	256,724	..	..	..	..	10?	72	..	..	256,796
1884	4,079	385	..	73,422	25,795	99,217	..	..	..	..	950?	7,600	..	..	106,817
1885	4,401	300	..	88,020	14,400	102,420	..	..	..	..	..	..	..	..	102,420
1886	235	6,678	24½	3,760	242,390	246,150	..	..	..	..	..	..	..	..	246,150
1887	7,491½	..	120 5/6	119,864	49,000	168,864	..	..	..	..	410	8,200	Mica 12 lbs.	240	177,304
1888	3,392	1,290	1,207¾	66,440	749,946	816,386	..	..	10 0	400	1,850	37,000	..	..	853,686
1889	2,306	761	1,343	46,120	310,250	356,370	..	..	..	..	7,530	64,000	..	..	420,370
1890	400	1,236	609	3,400	223,392	226,792	..	..	..	..	8,670	72,215	Antimony	1,200	300,207
1891	7,060	3,626	1,139	63,540	502,310	565,850	..	..	..	..	19,150	57,900	Do.	1,000	624,750
	283,355½	16,772½	4,441 1/6	6,967,043	2,226,747	9,193,790	319	29,604	2,853 11	119,804	38,570	247,087	144,050	6,540	9,594,717

THE CAPELL AND GUIBAL VENTILATORS CONTRASTED.

Mittheilungen über eine grosse Ventilations-Anlage nach dem Patente Capell auf Zeche Prosper I bei Berge-Borbeck. By M. KATTWINKEL. *Zeitschrift für das Berg-, Hütten- und Salinen-Wesen im Preussischen Staate*, 1890, *vol. xxxviii., pages* 347-349.

The Prosper I. pit, in which 1,000 workpeople are employed, was until recently fitted with a Guibal ventilator, with a diameter of 39¼ feet. This was found insufficient for the proper ventilation of the workings, and a Capell ventilator was put up instead. The new ventilator was tested in July, 1890. At 72 revolutions there was a mean air-velocity in the upcast shaft of 1,510 feet per minute, and the cross section being taken as about 52 square feet, the amount of air passing through per minute would be about 82,600 cubic feet.

In the downcast shaft, at the same number of revolutions, the average velocity was 1,790 feet per minute, giving, with a cross section of 59 feet, about 105,600 cubic feet per minute.

Further experiments were made at 80 and 87·5 revolutions, and the useful effect was found to lie between 51 and 53 per cent. of the steam-power used, or 7·6 times the useful effect obtained with the Guibal fan.

Attention is drawn to the lubricant used, which consisted of finely comminuted suet, over which a fine stream of water was made to trickle continuously; it was thus possible to keep the bearings quite cool, a *desideratum* unobtainable with the other lubricants (oil, tallow, etc.) used on previous occasions. O. S. E.

THE VENTILATING APPARATUS AT THE HEILBRONN SALTWORKS.

Die Grubenventilatoranlage auf dem Salzwerk Heilbronn. By FR. BUSCHMANN, 1893. *Zeitschrift des Vereines Deutscher Ingenieure, vol. xxxvii., pages* 29-34, *with figures in the text.*

The plant now in use was erected in the rock-salt mine at the beginning of 1892 at a moderate cost, and shows comparatively favourable results. The local conditions were such as to favour the use of a small, quick-running ventilator, and the author points out that the comparisons frequently drawn between large and small ventilators, to the detriment of the latter, are now proved to be untrustworthy.

The Pelzer ventilator works unilaterally, and is a combination of the helical (screw) and centrifugal ventilators, having straight and centric blades. This combination permits of a relatively large inlet with a fan of small dimensions. The conical fan-wheel has a diameter of 5⅓ feet, with a rim-breadth of 16 inches. The radially-arranged twelve blades are so placed that three pairs of diametrically opposed blades reach to the nave of the fan-wheel, while the six intervening blades are left short of it to allow space for the central inflow of air. These shorter blades are joined by means of a screw passing through the wheel-nave, so as to neutralize the pulling strain due to centrifugal power. The wheel-nave is prolonged backwards into a cone that meets the conical casing to which the blades are fixed. In front of the fan-wheel, and joined thereto by a sheet-iron cylinder, is the helical ventilator (diameter 5 feet, rim-width 11 inches), this also has twelve blades which are riveted to those of the fan-wheel and to the sheet-iron cylinder. The axle or shaft of the ventilator is 7⅓ feet in length, with an average diameter of 3 inches.

The ventilator is driven by the principal driving-shaft of the salt-grinding mill, the necessary power being transmitted thence by a manila-hemp rope which has been dipped in tar. The steam engine is a single-cylinder one (cylinder 21 inches in diameter, and 40 inches stroke).

Elaborate arrangements were made for the purpose of measuring the efficiency, etc., of the ventilator, and for the velocity-measurements a Casella anemometer was used, the constant of which is 10, and the time-limit selected for each measurement was 10 minutes. The experimental results are roughly expressed in tabular form as follows :—

No. of experiment	1	2	3	4	5
Revolutions of engine per minute.	40	45	50	55	60
Indicated power of engine (h.p.)...	8·505	10·598	12·956	15·505	18·460
Revolutions of fan per minute ...	184	206	228	249	270
Volume of air per minute (c. feet)	33,853	36,006	38,477	40,595	43,419
Water-gauge—					
Theoretical depression $H = \frac{yu^2}{g}$ (inches)	1·172	1·470	1·800	2·149	2·528
Observations—					
Depression h_1 (inches) ...	0·394	0·512	0·669	0·827	0·984
Compression h_2 (inches) ...	0·551	0·630	0·709	0·827	0·945
Water-gauge h (inches)... ...	0·945	1·142	1·378	1·654	1·929
Correction of water-gauge for velocity h_3 (inches)	0·193	0·220	0·252	0·279	0·322
Observed water-gauge $H_1 = h - h_3$ (inches)	0·752	0·920	1·126	1·375	1·607
Manometric efficiency $H_1 \div H$ ..	0·641	0·626	0·625	0·639	0·635
Power in the air (horse-power) ...	4·062	5·306	6·931	8·914	11·067
Mechanical efficiency	0·472	0·500	0·535	0·575	0·599

All the experiments appear to have been made on February 17th, 1892, between the hours of 3·52 p.m. and 7·30 p.m. The temperature aboveground at the time varied between 29 and 27 degs. Fahr., while in the airway at the ventilator it was uniformly 58 degs. Fahr.; and the height of barometer was 29·75 inches.

The total cost of the ventilator, including labour for putting it up, gear for transmission of power, and measurement appliances, was £120. The daily working expenses at 18 horse-power and 42,360 cubic feet per minute, including 5 per cent. for depreciation, do not exceed 2s. 9½d. M. W. B. and O. S. E.

A DOUBLE VENTILATING FAN.*

Verbundventilator. *By* C. Wenner. *Zeitschrift des Vereines Deutscher Ingenieure*, 1893, *vol. xxxvii., pages* 14-15. *With two figures.*

The ventilator herein described is apparently an improved form of an apparatus, in which two fans (instead of being concentrically arranged) are placed side by side and combined into one double fan. The two casings with their appropriately shaped blades are separated from one another, and closed up laterally by the solid disc carrying the blades. The double fan, being thus all of one piece, is driven by a single pulley or drum. Further, the double-walled fan-chamber is divided by the fan-disc into two compartments, an outer and an inner, and the air-current

* English patent, No. 13,725 of 1892.

takes the course which will be now described. Sucked in at one fan-inlet the air is thrown off at the periphery of the first fan at a certain pressure into the outer chamber through which it streams, around and outside the inner fan-chamber on the other side of the double fan. The air-current then enters through the inlet of the second fan, and at the periphery is thrown off into the inner chamber or casing, from which it passes to the conical outlet of the double fan.

The general formula for the depression or water-gauge produced by such a double ventilator is thus arrived at : Let

v = velocity of periphery of double fan ;
h_1 = water-gauge in outer chamber or casing ;
h_2 = water-gauge in inner chamber or casing ; and
k = a coefficient,

then, in an ordinary case where the ventilator sucks in air from the free atmosphere, the air-pressure produced by the first fan in the outer casing is expressed by the equation :—

$$h_1 = k \frac{v^2}{125^2}. \qquad (a).$$

That produced by the second fan, which exhausts air from the outer casing at the pressure h_1, being expressed by $h_2 - h_1 = k \frac{v^2}{125^2}$, consequently

$$h_2 = k \frac{v^2}{125^2} + h_1. \qquad (b).$$

Then, taking for the value of h_1 that given in (a), we obtain :

$$h_2 = k \frac{v^2}{125^2} + k \frac{v^2}{125^2} = 2k \frac{v^2}{125^2}. \qquad (c).$$

This value is exactly twice the depression or water-gauge that an ordinary ventilator, whose dimensions are one-half those of the double fan, would produce, other circumstances being similar.

The result of an experiment with a double ventilator, with a fan-diameter of 2 feet, at a speed of periphery of 102 feet per second, showed 6 inches of water-gauge at the conical outlet, the measurements being taken in the issuing air-current.

O. S. E.

VENTILATING FANS.

Les Ventilateurs de Mines : Rapport de la Commission instituée pour l'Étude des Ventilateurs, par la Direction de l'Industrie de la Société Générale pour favoriser l'Industrie nationale. *By* H. Mativa, E. Desvachez, I. Isaac, *and* N. Evrard. *Revue Universelle des Mines*, 1892, *series* 3, *vol. xx.*, *pages* 133-187, *and plates V.–VIII.*

The experiments detailed in the following table were made on four different types of mining fans—viz., the Guibal, Capell, Ser, and Rateau. Four experiments were made on each of these fans at different mines, and one failing, there are fifteen, which are detailed in the following table.

The Committee give the approximate cost of the different fans, including the steam-engine for driving, foundations, and building for covering, as follows :—Guibal, £1,120 ; Ser, £1,280 ; Capell, £1,380 ; and the Rateau, £1,200 ; and they summarize the four different fans thus. They are all efficient from many points of view, and possess good qualities, but the Rateau appears to possess advantages which place it first, as it produces a higher pressure, has a higher mechanical efficiency, and works regularly and silently.

B. D.

Results of Experiments on Fans.

Experiment Number.	Description of Fan.	Name of Colliery.	Years in Use.	Cost of Installation. Total.	Cost of Installation. Fan only.	Dimension of Fan. Diameter.	Dimension of Fan. Width.	Dimension of Fan. Diameter of Inlet of Fan.	Dimensions of Cylinders of Engine. Diameter of Piston.	Dimensions of Cylinders of Engine. Stroke.	Revolutions per Minute. Engine.	Revolutions per Minute. Fan.	Speed of Periphery of Fan per Minute.	Indicated Horse-power.	Sectional Area of Air drift where measured.	Air per Minute. Speed per Minute	Air per Minute. Volume per Minute.	Water-gauge. Observed.	Water-gauge. Calculated for Rim Speed of 6,888 Feet.	Theoretical Work of Air Displaced.	Equivalent Orifice.	Mechanical Efficiency.	Manometric Efficiency. Vacuum.
				£	£	Feet.	Feet.	Feet.	Inches.	Inches.			Feet.		Sq. Ft.	Feet.	Cu. Ft.	Inches.	Ins.	H.P.	S. Ft.		
1	Guibal	Levant du Flenu Chesnes	15	1,240	311	39·36	8·20	13·12	26⅜ Fan driven	33½ direct.	46 56 65	46 56 65	5,683 6,919 8,032	76·54 138·56 202·67	50·05	2,434 2,714 3,184	121,900 135,842 159,423	2·16 3·31 4·21	3·27 3·17 3·09	41·60 70·82 105·87	31·75 28·63 29·70	0·543 0·511 0·522	
																		Means	3·18		30·02	0·525	0·539
2		Marcinelle Nord	9⅓	1,200	256	....	....	....	24·40 Fan driven	39·37 direct.	50 60 70	50 60 70	6,180 7,417 8,647	59·10 88·51 150·03	43·04	1,210 1,421 1,631	59,125 61,048 70,262	2·87 4·09 5·79	3·53 3·66 3·57	23·61 39·66 64·09	11·72 11·62 11·19	0·399 0·448 0·427	
																		Means	3·586		11·51	0·424	0·607
3		Dampremy	5⅓	1,236	270	29·53	6·89	9·84	24·40 Fan driven	33·46 by belt.	40 47 53	64 75 85	5,931 6,950 7,877	57·69 88·32 116·48	70·37	879 976 1,021	61,941 68,733 71,908	3·07 4·01 4·84	3·970 3·697 4·130	29·98 46·14 54·89	13·56 13·13 12·48	0·520 0·522 0·471	
																		Means	3·94		13·02	0·504	0·667
4		Horloz à Tilleur	¾	1,120	180	19·02	6·40	6·89 Two Inlets	16·50 Fan driven	29·52 direct.	100 106 120	100 106 120	5,972 6,331 7,167	80·87 99·04 123·46	83·06	1,173 1,259 1,348	97,469 10,476 10,605	3·15 3·54 4·25	4·19 3·92 4·18	48·79 58·96 75·09	20·98 21·41 20·98	0·598 0·595 0·608	
																		Means	4·18		21·09	0·600	0·694
5	Ser	Davy	3½	720	249	4·59	0·79	2·79	11·81 Fan driven	25·6 by belt.	65 75 85	336 391 435	4,843 5,636 6,927	10·50 16·63 21·82	34·22	535 639 775	1,832 2,189 2,655	1·476 1·850 2·244	2·76 2·94 2·74	4·21 6·38 9·39	5·81 6·13 6·78	0·401 0·384 0·430	
																		Means	2·81		6·24	0·405	0·477
6		Dutemple	4½	820	300	5·25	0·92	3·11	13·9 Fan driven	29·5 by belt.	63 73 80	302 345 377	4,976 5,683 6,212	25·08 37·11 46·32	66·389	623 759	4,144 5,046 5,373	1·93 2·68 3·07	3·93 3·69 3·77	12·60 21·30 26·00	11·41 11·83 11·72	0·503 0·574 0·562	
																		Means	3·79		11·62	0·546	0·643

Results of Experiments on Fans.—*Continued.*

Experiment Number.	Description of Fan.	Name of Colliery.	Years in Use.	Cost of Installation		Dimension of Fan.			Dimensions of Cylinders of Engine.		Revolutions per Minute.		Speed of Periphery of Fan per Minute.	Indicated Horse power.	Sectional Area of Air-drift where measured.	Air per Minute.		Water-gauge.		Theoretical Work of Air Displaced.	Equivalent Orifice.	Mechanical Efficiency.	Manometric Efficiency, Vacuum.
				Total.	Fan only.	Diameter.	Width.	Diameter of Inlet of Fan.	Diameter of Piston.	Stroke.	Engine.	Fan.				Speed per Minute.	Volume per Minute.	Observed.	Calculated for Rim Speed of 6,388 Feet.				
				£	£	Feet.	Feet.	Feet.	Inches.	Inches.			Feet.		Sq. Ft.	Feet.	Cu. Ft.	Inches.	Ins.	H-P.	S. Ft.		
7	Ser	Moulin	7½	1,280	480	6·56	1·18	3·84	19·7 Fan driven	27·6 Fan by belt.	57 67 72	302 365 382	6,220 7,270 7,870	45·05 74·66 88·88	43·420 23·02¢	1,422 305 1,635 358 1,895 391	75,966 87,487 10,130	1·614 2·362 2·677	2·09 1·98 2·05	19·33 32·71 42·76	22·92 21·73 23·67	0·429 0·438 0·481	
																		Means	2·04		22·70	0·449	0·345
8		St. Mark	1	1,400	414	8·20	1·47	4·92	15·2 19·7 Tandem two cyldrs.	33·4 Fan driven by belt.	36 46 56	122 154 186	3,141 4,064 4,789	17·12 28·15 59·88	86·400	608 757 962	52,576 655,710 832,146	0·945 1·378 2·086	4·16 4·54 4·31	7·83 14·23 28·68	20·66 21·41 22·06	0·457 0·505 0·479	
																		Means	4·33		21·30	0·480	0·734
9	Capell	No. II. Prosper, Westphalia	2½	1,440	360	12·30	6·16	6·89 Two Inlets.	20·5 Two engines coupled.	31·5 Driven by belt.	46 56 66	184 224 264	7,106 8,651 10,196	63·54 93·78 148·08	58·53	1,231 1,525 1,699	72,157 89,348 99,590	3·38 4·33 6·10	2·74 2·78 3·18	38·51 60·98 95·79	15·06 16·47 18·38	0·606 0·650 0·647	
																		Means	2·90		15·6	0·634	0·491
10		Kœnigin Elizabeth	2½	700	210	8·20	5·90	4·59	15·75	23·6 Driven by belt.	40 50 60	160 200 240	4,120 5,152 6,183	6·60 14·63 24·62	39·06	704 940 1,041	27,535 36,762 40,697	1·06 1·8 2·52	2·97 3·38 3·12	4·63 10·82 16·16	10·22 10·33 9·47	0·701 0·739 0·656	
																		Means	3·16		10·01	0·698	0·534

RESULTS OF EXPERIMENTS ON FANS.—*Continued.*

Experiment Number.	Description of Fan.	Name of Colliery.	Years in Use.	Cost of Installation.		Dimension of Fan.			Dimensions of Cylinders of Engine.		Revolutions per Minute.		Speed of Periphery of Fan per Minute.	Indicated Horse-power.	Sectional Area of Air-drift where measured.	Air per Minute.		Water-gauge.		Theoretical Work of Air Displaced.	Equivalent Orifice.	Mechanical Efficiency.	Manometric Efficiency, Vacuum.
				Total.	Fan only.	Diameter.	Width.	Diameter of Inlet of Fan.	Diameter of Piston.	Stroke.	Engine.	Fan.				Speed per Minute.	Volume per Minute.	Observed.	Calculated for Rim Speed of 6,888 Feet.				
				£	£	Feet.	Feet.	Feet.	Inches.	Inches.						Feet.	Cu. Ft.	Inches.	Ins.	H-P.	S. Ft.		
11	Capell	Dahlbusch	½	1,380	300	11·81	5·25	7·216 Two Inlets.	17·7 Two engines coupled. Driven by belt.	23·5	36 40 44	160 178 196	5,931 6,598 7,263	30·30 40·54 52·68	46·16	881 997 1,039	40,722 46,086 47,963	2·32 2·83 3·11	3·09 2·80 3·13	14·91 20·68 23·53	10·22 10·44 10·22	0·492 0·508 0·447	
																		Means	3·01		10·22	0·482	0·509
13	Rateau	Villebœuf (Soufflant)	1¼	1,000	420	6·56	·52	3·93	12·8	19·7 Driven by belt.	100 120 125	177 216 255	3,644 4,447 5,252	19·55 32·81 35·74	30·88	1,584 1,702 1,877	48,955 52,604 58,015	1·22 1·81 1·89	4·33 4·35 3·25	9·41 15·01 17·28	17·00 14·96 16·46	0·481 0·457 0·483	
																		Means	3·97		16·14	0·473	0·673
14		Villars	$\frac{5}{8}$	1,000	380	6·56	·52	3·93	15·0	23·6 Driven by belt.	110 120 130	237 264 285	4,880 5,437 5,870	23·82 30·72 40·37	37·01	1,291 1,350 1,527	47,823 50.005 56,562	2·64 3·19 3·74	5·14 5·11 5·24	19·88 25·13 33·35	11·30 10·76 11·19	0·839 0·818 0·826	
																		Means	5·165		11·08	0·827	0·875
15		Rien du Coeur Quaregnon	½	1,200	480	9·18	·754	5·51	29·5	47·2 Driven by belt.	35 45 55	145 186 228	4,179 5,362 6,573	30·79 49·94 85·57	37·98	1,021 1,225 1,602	38,822 46,615 59,488	1·77 2·83 4·29	4·82 4·65 4·74	10·77 20·83 40·24	11·19 10·65 11·08	0·350 0·417 0·470	
																		Means	4·74		10·97	0·412	0·802
16		Montrambert de la Beraudière	$\frac{5}{16}$	1,200	580	9·18	·735	5·51	15·0	23·6 Driven by belt.	80 101½ 114	115 150 170	3,316 4,325 4,902	16·05 31·83 41·12	61·12	881 1,115 1,233	53,925 68,249 75,462	1·30 2·05 2·68	5·19 5·60 5·28	11·04 22·02 31·85	18·07 18·29 17·65	0·687 0·692 0·775	
																		Means	5·35		17·97	0·718	0·907

THE UNDERGROUND VENTILATOR AT THE HANSA COLLIERY.

Die unterirdische Ventilatoranlage der Zeche Hansa. ANON. *Glückauf*, 1892, *pages* 909-911, *with two plates.*

Most of the fresh air passes down No. 2 shaft to the 664 metres level, the rest of it going on to the 700 metres level, and after travelling through the workings above those levels, it reaches by means of a blind shaft and a rise-drift the airway, and thence the ventilator.

The ventilator is placed near the No. 1 shaft, and is driven by a steam-engine with two cylinders, each 16 inches in diameter and 2 feet stroke. The power is transmitted from the engine to the ventilator by means of a belt geared $3\frac{1}{2}$ to $1\frac{1}{2}$.

The ventilator is placed between two walls of masonry, which are strengthened by iron clamps. Between these two walls is fixed a dome or casing of sheet-iron, forming the upper portion of the comparatively broad spiral-shaped fan-chamber. The spiral casing is continued below the ventilator, broadening out into the masonry, and ending ultimately into the discharge airway.

There are three air-channels between the ventilator and the engine-room; that of largest diameter, communicating with the suction-inlet of the ventilator, takes up the hot foul air from the engine-room, whilst the other two bring the fresh air from the downcast pit. O. S. E.

MOT SYSTEM OF WINDING BY ENDLESS-ROPE.

Étude sur le Cable de Mot. By J. DOURY. *Le Génie Civil*, 1892, *vol. xxi., pages* 67-71, *and* 11 *figures.*

The installation described was erected at one of the shafts of the Marles collieries, Pas de Calais, in 1888. Previously, winding had been effected by flat aloes ropes, with wooden guides, and it was desired to increase the output without increasing the number of boilers in use.

Iron guides were used similar to those first adopted at Mariemont, consisting of Vignole rails, weighing 70 lbs. per yard (35 kilogrammes per metre), fastened to transverse beams dividing the shaft through the centre. Two cages run on either side of the central division, there being two guides to each cage.

In the Kœpe system, the cages are connected to the ends of a single head-rope, which passes over two pulleys at the pit-heads and around a drum worked by the winding-engine. The cages are similarly joined below by a tail-rope, the length of which is equal to the depth of the shaft. The main objection to the system is that of accentuating the gravity of an accident, as in case of a breakage of the head-rope both cages would be precipitated down the shaft.

Mr. de Mot improved this system by duplicating both head and tail-ropes, so that not only do they give, combined, double the strength necessary, but the improbability of a simultaneous breakage renders accident almost impossible; for, in the case of a single rope breaking, the remaining rope, with the aid of the safety-rope, would enable the winding to proceed. In order to obviate difference of lengthening, both head and tail-ropes themselves form an endless rope.

The head-rope consists of two round steel ropes joined at the corresponding extremities by connecting chains which work round loose pulleys at the top of the cages, so as to correct the difference of lengthening. At the same time, they are also connected by safety-chains to keep the cages from falling in case of breakage. The wire ropes used are 1·30 inches (33 millimetres) in diameter, weigh 8·8 lbs. per yard (4·360 kilogrammes per metre), and cost 7d. per lb. (1·65 francs per kilogramme).

The tail-rope is a flat iron rope, 3·74 inches by 0·98 inch (95 by 25 millimetres). The weight is 14·3 lbs. per yard (7·09 kilogrammes per metre) and the cost 4·3d. per lb. (1 franc per kilogramme).

The diameter of the drum is 14·76 feet (4·50 metres), but would, with advantage, have been smaller, from 12 to 13 feet (3·80 to 4 metres), in order to diminish the work of the machine in overcoming inertia when starting. The total weight was 5½ tons (5,645 kilogrammes), and the cost £150 (3,836·51 francs).

The total cost of the installation was £1,580 (39,938·50 francs).

At Marles, the increase in rapidity of winding was marked, but not so great as would have been the case with a deeper shaft. The time occupied in changing the tubs was greater than would have been necessary had the landing-places been suited to the system.

Besides the advantage of safety in case of breakage, the system, of course, entirely removes the possibility of over-winding. On the other hand, it is not suited for use where it is required to wind from various levels.

The author considers that for shafts of from 400 to 800 yards deep, where the extraction is only from one level, the system would prove very advantageous.

G. E. C.

BRINE SPRINGS IN THE ARGENTINE REPUBLIC.

Las vertientes de agua salada de Tapias. *By* EUGENIO TORNOW. *Anales de la Sociedad Cientifica Argentina, Buenos Aires,* 1892, *vol. xxxiv., pages* 206-208.

About 3¾ miles to the eastward of Tapias station, on the North Central Railway, province of Tucuman, the springs gush out in a ravine which marks the course of a dried-up burn. In the old stream-bed trickles a tiny rivulet of brackish water proceeding from other springs more to the westward, and the water from all the springs uniting into one streamlet, the whole pours itself a short distance off into the Rio Salè.

The springs visited by the author make their way through a stratum of sand and bluish-grey pebbles, and then through about 6½ feet thick of reddish clay. No. 1 on the left bank of the ravine yielding 1·98 gallon per minute, and No. 2 on the right bank 2·64 gallons per minute. The former, which the landowner has surrounded with a stone enclosure, shows a temperature of 85·4 degs. Fahr., and the latter, issuing freely, a temperature of 77·9 degs. Fahr. On analysis the water yields an average of 19·4 per cent. of salt (sodium chloride), 1·73 per cent. of sodium sulphate, 0·2 per cent. of calcium sulphate, 0·95 per cent. magnesium sulphate, the total solid residue in 100 parts by weight of water being about 21½ parts. The rainfall of the country appears to have no effect whatever on the saltness of the springs. O. S. E.

Extracted from "Coal field."

GEOLOGICAL MAP
OF THE
IGARENI COAL-FIELD.
ATE OF HYDERABAD, INDIA)

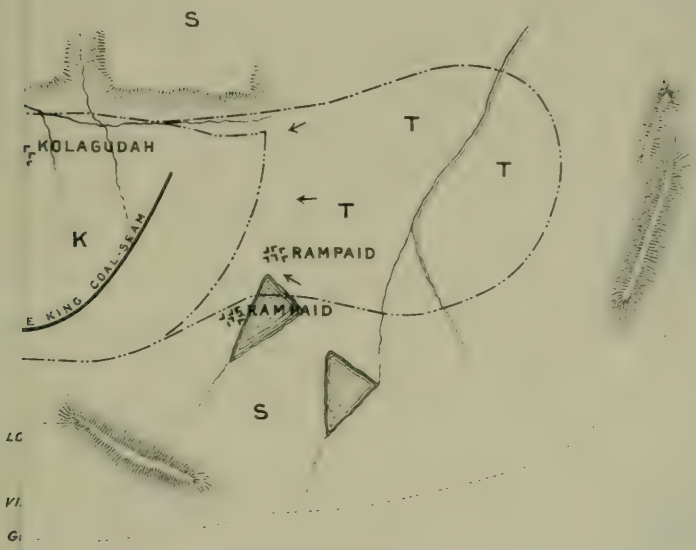
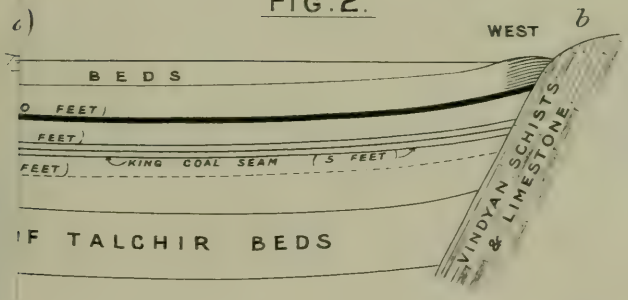


FIG. 2.



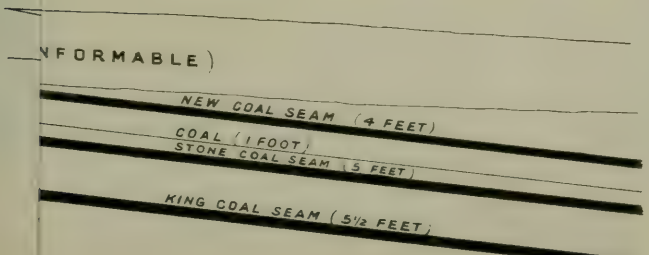


FIG. 1.

GEOLOGICAL MAP OF THE SINGARENI COAL-FIELD. (STATE OF HYDERABAD, INDIA)

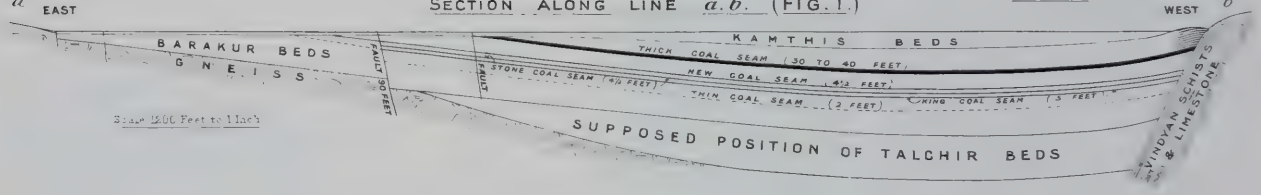
REFERENCE

TRIASSIC FORMATION
LOWER GONDWANA
K — KAMTHIS
B — BARAKUR, COAL-BEARING
T — TALCHIR
VINDYAN
S — METAMORPHIC SCHISTS
GNEISS
G — GNEISS

Scale 1½ Miles to 1 Inch

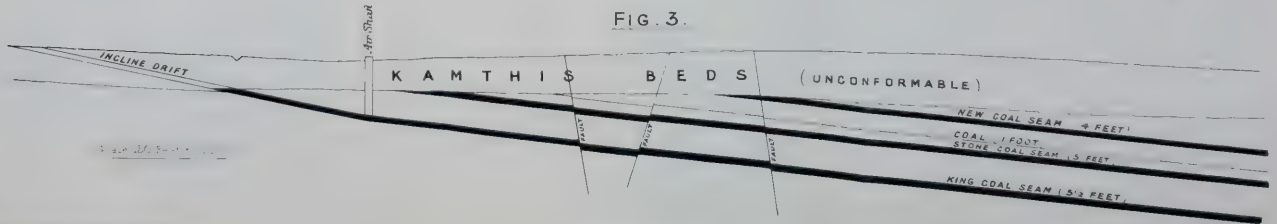
FIG. 2.

SECTION ALONG LINE *a.b.* (FIG. 1.)



Scale 1200 Feet to 1 Inch

FIG. 3.



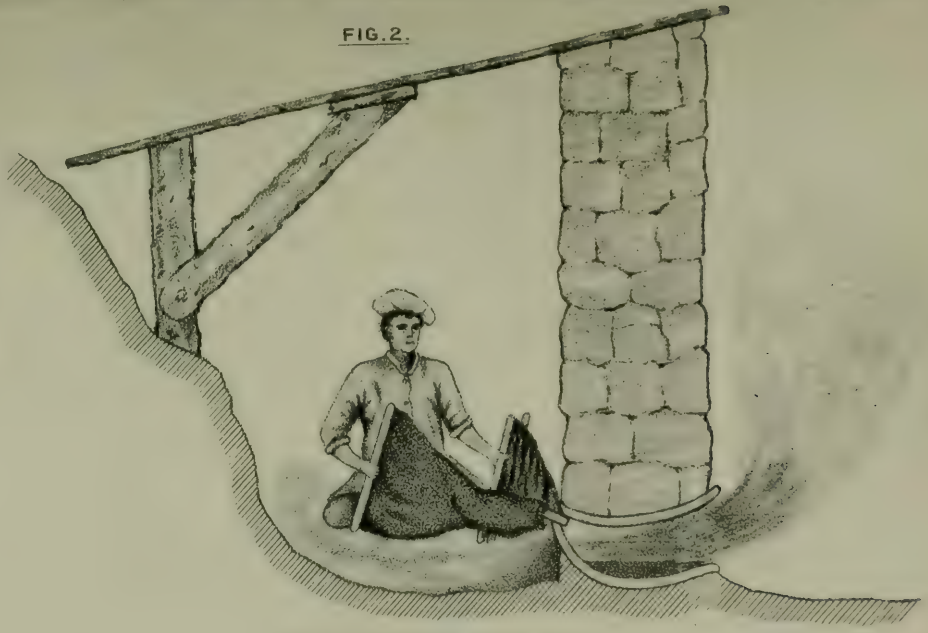
Federated Institution of Mining Engineers
Transactions 1903-4.

To illustrate Mr. A. L. Collins' Paper on "The Ghorband Lead-mines, Afghanistan".

Vol. VI. Plate XIII

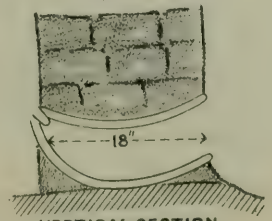
FIG. 1.

FIG. 2.



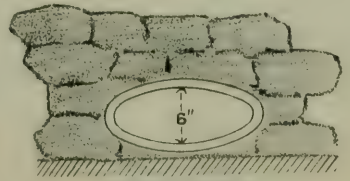
SIDE ELEVATION (PART IN SECTION)

FIG. 3.



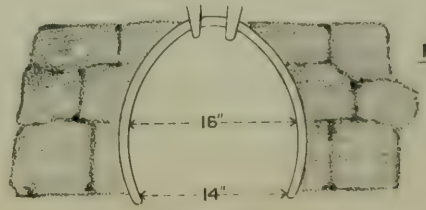
VERTICAL SECTION.

FIG. 4.



FRONT ELEVATION.

FIG. 5.



PLAN.

North Staffordshire Institute of Mining and Mechanical Engineers
Transactions 1903-4.

Vol. XLIII. Plate IV.

III.—NOTES OF PAPERS ON THE WORKING OF MINES, METALLURGY, ETC., FROM THE TRANSACTIONS OF FOREIGN SOCIETIES AND FOREIGN PUBLICATIONS.

THE DOURO ANTIMONY DISTRICT, PORTUGAL.

La Industria del Antimonio en Portugal. Anon. *Revista Minera*, 1894, *vol. xlv., page* 54.

The Douro metalliferous zone extends in a north-easterly direction from Castello de Paiva to Vallongo, with a length of 37 and a width of 6 miles. The rocks are argillaceous, talcose, and micaceous schists, grauwackes and quartzites of Laurentian, Cambrian, and Silurian age. The veins traverse these rocks in all directions, the gangue consisting of white quartz and sometimes of silicified fragments of country rock, carrying stibnite, and generally also some gold in the quartz. The principal mining centres are Vallongo and Gondomar. G. E. C.

THE ASBESTOS FIELDS OF PORT-AU-PORT, NEWFOUNDLAND.

By C. E. Willis. *Mining Society of Nova Scotia, vol. ii., page* 166.

The metamorphic rocks and serpentines of Quebec dip under the Gulf of St. Lawrence to reappear on the western coast of Newfoundland, and extend probably across the island. The district, situated at and near the eastern coast of Port-au-Port bay, near the southern boundary of the serpentine rock, is seamed in many places with asbestos, and contains veins of copper and specular iron. Only surface work has been done so far; but the asbestos fibre is of good quality, up to 2½ inches long, and practically identical with the Canadian product. G. E. C.

A COAL CALORIMETER.

By Geo. H. Barrus. *American Society of Mechanical Engineers,* 1893, *vol. xiv., page* 816.

The Barrus calorimeter consists of an outer glass vessel filled with water, and an inner and smaller bell-shaped vessel with a perforated base and a long glass neck, rising several inches above the water in the outer vessel. A crucible made of platinum is placed inside the inner vessel, and oxygen is introduced into it from a reservoir by means of a tube passing through the neck of the glass or combustion-chamber. In making a test 2 kilogrammes of water are used, and 1 gramme of pulverized coal, and the gases of combustion rising from the crucible escape downwards through the perforated base, and bubble up through the water in the outer vessel. Great care is taken to bring the temperature of the water and of the instrument approximately to that of the room, and thus obviate the necessity of allowing for radiation. In most coals the temperature of the water is raised by combustion from 5½ degs. to 6½ degs. Fahr.

[The Barrus calorimeter seems very similar to the glass apparatus designed by Professor Thomson of Manchester, and well known in England.] B. D.

THE FORMATION OF COAL *IN SITU.*

Eine gewöhnliche Art der Erhaltung von Stigmaria als Beweis für die Autochthonie von Carbon-Pflanzen. *By* H. POTONIÉ. *Zeitschrift der Deutschen geologischen Gesellschaft*, 1893, *vol. xlv., pages* 97-102, *figures in text.*

Speaking from his own observations in Upper Silesia, the author notes the extraordinary abundance of *Stigmaria ficoides* in the clay-slate [? fire-clay] under such conditions as to make it plain that the plant has been fossilized where it grew, and therefore that the "clay-slate" is, in a certain sense, a "fossil soil." He figures a specimen of *Stigmaria* with the appendices ramifying on three sides from corresponding "scars" on a central trunk. Enough of their botanical anatomy is now known, enough is known of their feeble resisting capacity to mechanical forces, to feel assured that the plants could not have been transported, say drifted, in their present state, unless the mass of surrounding soil was carried bodily with them. Now, such specimens as these occur in Upper Silesia in vast numbers. There is, indeed, reason to believe that a large portion of the "clay-slate" [? fire-clay] "parting" of the Coal-measures in that area must have been laid down before the *Stigmariæ* grew in it, and the occurrence of coal-vegetation *in situ* is thus shown to be far more extended than the partizans of the "mainly drift-coal" theory are prepared to admit. O. S. E.

BROWN COAL IN GALICIA, AUSTRIA-HUNGARY.

Ueber die Braunkohlenvorkommen in Oesterreich-Ungarn. *By* PROF. R. H. *Glückauf*, 1893, *vol. xxix., page* 1391.

Brown coal is found in several places in Austria-Hungary. In West Galicia there is a 23 feet seam at Tarnou composed of 4 feet of slaty brown coal, while the upper part consists of good quality bright brown coal. The seam has an east-and-west strike, and has been proved over a distance of $\frac{1}{4}$ mile. The depth of the workings is about 150 feet. A smaller coal-seam, 3 feet in thickness, of inferior quality is found at Neu Laudec on the Dunajec river and at Niskova.

In the Cretaceous formation some small seams of lignite are found. In places there are three seams varying from $1\frac{1}{2}$ to 3 feet in thickness, but usually there is only one seam of from 3 to 6 feet. The lignite contains 6 to 12 per cent. of ash, and contains 12 per cent. of water.

In the south-east of Galicia there is a coal-seam, 18 inches thick, extending about 13 miles in length. The coal is a hard black brown coal (glanzkohle), which breaks into long pieces like wood. These coal-mines are chiefly worked by adits. There are altogether ten brown coal-mines in Galicia, but only five are now working, employing about 340 workmen. W. F. W.

THE CARPANO COAL-FIELD, ISTRIA, AUSTRIA.

Die Kohlenmulde von Carpano in Istrien. *By* Dr. K. ANT. WEITHOFER. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, *vol. xli., page* 261 *and* 275, *and plate XII.*

This little-known coal-field is situated in Istria, on the promontory of Quarnero in the Adriatic sea, between Trieste and Fiume, 21 miles from the railway-station of Pisino. It is near the northern extremity of the Gulf of Arsa, and is connected by a

small narrow-gauge railway with the harbour of Valpidocchio. It is about 4 miles from the little seaport of Rabaz, and is the only coal-mine now being worked in Istria.

The Coal-measures form part of a geological basin or depression stretching from Trieste across Istria. The formation is of the older Tertiary period, and rests upon Hippurite limestone of the Upper Cretaceous formation. The depression is filled with nummulitic limestone, compact sandstone, and argillaceous marl, below which is found a mixture of marine and coal-bearing lacustrine sediments, called the Cosina beds, and forming part of the Liburnian stage of the lower Eocene period. It is of special geological interest as showing the alternations of marine and fresh-water formations, the sea having periodically advanced and retreated along the shores of the basin.

The Liburnian series may be divided into three periods:—First, the lower foraminiferal stratum, or deposition of limestone in a deep sea; next, the Cosina beds, characterized by a fresh-water fauna and flora, when the sea retreated, and was replaced by fresh or brackish water vegetation; and, lastly, the upper foraminiferal limestone, representing another advance of the sea and the purely marine formation of nummulitic limestone.

The coal-seams are found in the middle or Cosina beds, especially in the neighbourhood of the town of Albona, near which the Carpano-Vines mine is situated. Coal is present elsewhere in the Cosina beds of this Istrian basin, but not in sufficient thicknesses to repay the cost of working.

During the period when the sea covered the greater part of northern Istria, the lowest stratum of limestone, in this south-eastern district, was worn into great hollows, the irregular depressions and valleys were filled up with successive layers of brown and black fresh-water bituminous limestone, and numerous coal-seams formed. The thickness of the limestone varies from a few inches to 9 or 10 feet, and the coal from a thin layer to a seam, sometimes 6 or 7 feet thick, but the line between the coal and the surrounding rock is always sharply drawn.

The value of the coal is much diminished by the frequent occurrence of what is called "sootiness" in the seams. The coal loses its firmness and consistency, becomes friable, and when dry resembles soot; it is usually wet and then forms a kind of black grease. From the way the "soot" makes its appearance, it must be attributed to the oxidizing influence of the atmosphere. All the upper seams, even where they rest upon the limestone, have a belt of this oxidized coal towards the edge of the depression, and wherever clefts and fissures occur, produced by the water oozing in at the side of the strata. Even where the fissure is closed, it is possible to trace its former presence in the seam, the sooty coal in two or three places being found in the midst of the coal-seam.

The proportion of water in this decomposed coal is 1·70 as against 1·46 per cent. in the coal, and of ash 14·96 as against 8·84 per cent. in the coal; the other chemical constituents diminish in inverse ratio, especially the carbon and hydrogen.

The coal seldom contains well-defined remains of plants, but in the neighbourhood of Pisino fossils of various tropical shrubs have been found, such as now flourish principally in Australia and South Africa. The nearest parallel to the state of rank vegetation depicted in the Coal-measures of South Istria are the huge cypress swamps of North America. If, in a depression such as we have described, there were successive periods of high water with depositions of limestone, and low water with rich marsh vegetation, these would explain the processes of building up the Carpano basin.

The Arsa coal differs in several respects from ordinary lignite or brown coal. It is shiny, almost black, and has a slaty appearance in the limestone seams, and a

seamy texture in the roof seams, with uneven fracture. When rubbed it emits a strong bituminous smell, and burns with a bright flame and much smoke. Analyses of its chemical composition have been made by Prof. Schwackhofer as follows :—

Carbon	60·94
Nitrogen	1·28
Hydrogen	4·08
Water	2·01
Oxygen	17·54
Ash	14·15

In its large proportion of carbon it resembles Bohemian (hard) coal, and it has a high calorimetric value, and low percentage of water. The latter is one of the chief differences between black and brown coal, and is usually from 15 to 20 per cent. in lignite, and from 4 to 6 per cent. in black coal. As this low proportion of moisture stands in inverse ratio to the density of the coal, great compression and hence a change in the condition of the fossil combustibles must have taken place in the Cosina beds. The capacity of coal for absorbing moisture is closely connected with its assimilation of oxygen, and in the Carpano coal the latter is correspondingly small.

It has been shown that the spontaneous combustion of many coal-seams may be ascribed less to their proportion of iron pyrites, than to the amount of oxygen which they absorb. A coal containing much pyrites and slight capacity for absorption is said to decompose more rapidly under the influence of moisture, and kindle more readily than when dry. Accordingly, the coal in the Carpano mine has no tendency to become heated, and it is only the refuse heaps aboveground, and those very rarely, which take fire.

When mixed with potash or carbonate of soda the Carpano coal, even after being heated to the boiling point several times, still retains a pale brown colour. Treated with a solution of bi-chromate of potash and sulphuric acid, for fourteen days and prolonged heating, it yielded black coal-dust, burning with a flame like ordinary coal. Brown coal with this solution usually loses its colour after a few hours, and has no combustible residuum, while black coal after dissolving for several months in chromic acid, retains its black combustible character. The best result was obtained with chlorate of potash, nitric acid, and ammonia. In this and other respects the most striking characteristic of the Carpano coal is its resemblance to pure black (hard) coal. It has a great tendency to cake, and yields about 55 to 60 per cent. of coke, which has been utilized in the furnaces in the neighbourhood. For blast furnaces, however, the percentage of sulphur and ash is probably too high to make it available. The mine is worked by galleries 19 to 26 feet broad, leaving pillars from 60 to 100 feet square.

There is a factory at the old harbour of Stallie for utilizing the dust coal in the manufacture of briquettes. These are made in presses with vertical rollers, pitch being used to bind the brick. About 17,000 tons were manufactured in 1891, and sent to Dalmatia, Trieste, Fiume, and Italy.

The number of miners in 1891 was 1,169; and the total coal production was 81,028 tons, for which a ready market was found in Istria, Dalmatia, and Italy.

B. D.

COAL-MEASURES IN POCKETS, BELGIUM.

Le Terrain Houiller de Bouge et de Lives. By X. STAINIER. *Annales de la Société Géologique de Belgique,* 1893, *vol. xx. (Mémoires), pages* 133-139, *and figures in text.*

The occurrences described in this paper were observed at Bouge and at Lives, where there are two small patches of Coal-measures north of the great Basse Sambre coal-field.

At the first-named locality the Coal-measures, instead of appearing in their regular order above the *Productus-giganteus* limestone Vg, are found between the two lower limestones Vf and Ve. In explanation of this abnormal sequence, the several hypotheses of folding and of one or more faults are discussed, but dismissed because they do not fit in with the facts. The author surmises that the deposit is a "pocket" of Coal-measures, contemporaneously formed in the underlying Carboniferous limestones, and in many respects analogous to the pockets of Tertiary sands and clays frequent in Belgium. A cavity being formed by dissolution in the limestones, so as to constitute a "natural shaft" or swallow-hole, the insoluble Coal-measures would sink into the cavity. Denudation subsequently planing off the upper portion of the "shaft" with its walls of limestone and overlying measures, the residual "pocket" would have all the appearance of a mass intercalated between two beds of limestone. Another circumstance adds to the probability of this explanation: the Coal-measures at Bouge are contorted in all directions, losing entirely their characteristic regularity of stratification, and this helps one to realize how the soft material slid and sank into the swallow-hole.

At Lives there is a precisely similar occurrence, and the author accounts for it in the same way. It will be seen that the paper is suggestive, because it affords a possible clue to many "puzzles in stratification." O. S. E.

BELGIAN CANNEL COALS.

Sur quelques Roches combustibles belges assimilées ou assimilables au Cannel coal anglais. By AD. FIRKET. *Annales de la Société Géologique de Belgique*, 1893, *vol. xx.* (*Mémoires*), *pages* 107-110.

The author has examined and determined the proportions of volatile matter, fixed carbon, and ash in five specimens of so-called cannel coal from as many Belgian collieries. The specimens from La Haye and Bois d'Avroy—where certain small seams have, with little apparent justification, been termed "cannel coal" for a great many years—he dismisses as "bituminous shale." Moreover, a specimen of the "cannel" from Horloz proves to be an anthracitic coal verging on true anthracite. From the Fanny pit of the Espérance colliery, Seraing, and the St. Léon pit, Mariemont, come true cannel coals: the Mariemont variety approaches jet in its susceptibility to polish and its applicability to ornamental purposes. Jet of a similar character has been found in the Haut Flénu and Couchant du Flénu collieries.

The discussion on this note will be found in pages 103-105, vol. xx. of the *Bulletin*. O. S. E.

THE HAINAUT CENTRAL COAL-FIELD, BELGIUM.

Étude sur la structure du bassin houiller du Hainaut dans le district du Centre. By ALPH. BRIART. *Annales de la Société Géologique de Belgique*, 1893, *vol. xxi. (Mémoires), pages* 125-149, *and plate III.*

The Central (*Centre*) coal-field lies towards the northern limit of the basin of Hainaut. Mining operations have for long been carried on in the northern part of the district, and the strata are found regularly dipping towards the south. This

part is designated the North Central (*Centre-Nord*). Farther south the strata are disturbed and irregular. Collieries have been recently sunk to the south of this disturbed region. The strata are still disturbed, but the southern dip is more constant. This part is known as the South Central (*Centre-Sud*).

There are thus two areas where coal is worked, the North and South Central, separated by a zone of greatly disturbed strata where coal has not been worked.

Messrs. Edm. De Cuyper and P. Dubar have arrived at the same conclusion, viz., that there are two distinct series of semi-bituminous seams. In the Northern and Central parts of the basin the two series are found lying one above the other. And as on stratigraphical grounds there is not space for both in the Southern parts, Mr. De Cuyper has to admit the disappearance of one without being able to explain how it so happens. If there are really two series the other must be accounted for.

Mr. Dubar applies the law that the percentage of fixed matter augments with the depth of the seam in the series. This rule is found true for a part of the basin only. For in the North Central field, on the supposition of two series, anomalies in the percentages are found. Thus to take a particular case, the fixed carbon increases from 65 per cent. in the higher seams to 89·37 at about 1,840 feet in depth. It then decreases to 81·99, to increase again to 88·41 per cent. Thus at depths below 1,840 feet the coal contains a percentage of fixed matter which, according to the rule, should belong to seams much higher in the series. A similar state of things is found in other parts. In spite of this, however, Mr. Dubar adheres to the view that there are two independent series. Mr. De Cuyper arrived at the same conclusion. Both, then, recognize two series of semi-bituminous beds in the Central district, and this, indeed, has been the opinion long accepted.

Mr. Briart believes that the supposed two series are really one, and that it is towards the east of the district, and especially at the Sart-le-Moulin colliery that the solution of the problem must be sought.

The conclusion reached by a careful study of this district is that the supposed two series are really only a repetition of the same series. On this supposition the anomaly in the percentages of fixed matter discovered by Mr. Dubar is explained, and the difficulty of the disappearance of the series of Mariemont requires no explanation.

This result has been brought about by a fault which is the prolongation of one long known in the province of Namur (where it attains its maximum at Vedrin) under the name of the St. Marc fault. There the coal-basin is separated into two bands by a mass of Carboniferous Limestone. This limestone has been brought up on to the coal by a fault inclined to the south. This fault can be traced west by superficial indications until it passes through the Central district. It is probably farther prolonged to the west as far as the north of the Mons district. By it the two areas of the North and South Central have been divided by a zone of greatly disturbed strata.

The task of identifying particular seams in the two series is a more difficult one. It is considered that the Allaye and Grande Veine coal-seams of the North Central are the same as those of Berlette and Grand Berlan of the South Central field. The Broze seam of the latter is also probably the d'Argent seam of the former. Further, the series of beds below the Fulvie-Hugo seam of the South Central represent those between the d'Argent and Gigotte seams of the North Central district. Thus, the Central coal-field of Hainaut, long quoted as an example of regularity, is probably, taken altogether, the most disturbed in Belgium. G. W. B.

FOLDING AND FAULTING OF THE BELGIAN COAL-FIELD.

Géologie des Environs de Fontaine-l'Evêque et de Landelies. By ALPH. BRIART. *Annales de la Société Géologique de Belgique*, 1893, *vol. xxi. (Mémoires), pages* 35-103, *and plates I. and II.*

The latest views of Belgian geologists on the interpretation of the extremely complicated structure of a part of their Coal-measures is as follows :—That the older rocks have been brought up and slidden over the younger by a remarkable series of overthrust faults.

The axis of the comparatively narrow band of ancient rocks, known as the *Crête du Condroz*, which lies a little to the south of the rivers Meuse and Sambre, seems always to have been the zone of least resistance in Belgium throughout geological time. In other words, the rocks of this region have experienced more disturbance than elsewhere in the country. The gigantic faults and overturnings of the rocks thus produced are of greater importance, and are better known, as this region lies in the midst of the Belgian Coal-measures.

The strange and anomalous position of Carboniferous Limestone and Devonian rocks above the Coal-measures has for long been a puzzle to miners and geologists. Various explanations have been offered. It has been suggested that the older rocks represent ancient cliffs round which the Coal-measures have been deposited, or that the older have been thrust up through the younger formations *en masse*. Various systems of gigantic faults have also been called into requisition. Extended geological investigations, and continued mining operations, however, have shown these explanations to be insufficient.

The presence of areas of Carboniferous Limestone and Devonian rocks (*lambeaux de recouvrement ou de refoulement*) above the Coal-measures may be accounted for by a series of reversed faults of low hade, known as overthrust faults, and along the plane of such faults the older are slidden over the younger rocks, and subsequent denudation has isolated these masses from the rest of the formation of the same age. In Belgium, in fact, is found a series of faults similar to that existing in the north-west of Scotland, where it has been shown by the Geological Survey that masses of Archæan gneiss have been thrust by similar faults over rocks of Silurian age.

The phenomena of this Belgian district may be the result of four principal thrusts from south to north following each other. The faults which have produced these results are of a peculiar nature. They do not run in straight lines, and are more or less nearly horizontal. They become more nearly horizontal with depth, and crop out to the south at almost the same altitude as to the north. Thus the plane of a fault cuts out a basin-shaped mass of strata. The fault *de la Tombe* is the result of the first great movement, and is the most northern of the series. Before this great disturbance took place, the coal-field, broadly speaking, was an immense monocline or basin, of V form, inverted towards the north. A fracture took place at a certain height in the strata of the southern edge of the basin. The upper part, following the plane of this fracture, was pushed to the north, while the lower strata remained in place. Thus the fault *de la Tombe* was produced. A part of the strata advanced as far as the northern edge of the basin, where Lower Coal-measures were left resting on Upper Coal-measures. Next, a fracture was produced in this transported mass. By it the strata were again pushed north, but the most northerly part of that originally transported was left finally arrested. Carboniferous Limestone strata were then pushed over Coal-measures. This forms the fault of *Fontaine-l'Evêque*. Towards the south these faults join, and have one thrust-plane.

A second fracture was produced in the first transported mass, and another thrust from the south brought the Lower Carboniferous Limestone and Devonian over the

Upper Carboniferous Limestone transported by the preceding movement. This forms the fault of *Leernes*. Again towards the south this fault has the same thrust-plane as the two preceding. The last manifestation of south-to-north movement is the southern fault. By it, on the southern edge of the basin, the Lower Devonian has been thrust over Coal-measures, Carboniferous Limestone, and Upper Devonian. Numerous foldings of the strata add to the complexity produced by the above faults. The most recent Coal-measures do not exist in Belgium, and it was while these were being deposited elsewhere that the above dislocations began to take place.

G. W. B.

ORIGIN OF THE AUTUN BOGHEAD.

Note sur la Formation schisteuse et le Boghead d'Autun. *By* B. RENAULT *and* C. E. BERTRAND. *Bulletin de la Société de l'Industrie Minérale, series* 3, 1893, *vol. vii., pages* 499-550, *and plate XXIII.*

The bed of boghead enclosed in the bituminous schists of Permian age forming part of the Autun coal-basin is extremely regular, and about 10 inches thick. The composition is as follows :—

Mineral Matter—Water	0·55		
Carbonate of lime	10·50		
Carbonate of magnesia	traces.		
Phosphate of lime	1·50		
Pyrites	1·00		
Clay and sand	26·30		
		39·85	
Organic Matter—Carbon	52·37		
Hydrogen	6·66		
Oxygen and nitrogen	1·15		
		60·18	
Total		100·03	

The organic contents are practically identical with Australian boghead, which, however, contains less mineral matter.

Examined microscopically, it proves to be composed of thalli of an alga, to which the name *Pila bibractensis* has been given, connected by an earthy substratum which the authors call the "fundamental matter." It contains coprolites, probably the fossilized excreta of fish inhabiting the lake, the ancient bed of which the boghead represents; and also siliceous concretions, often enclosing fossilized thalli, fish-scales, etc.

The conditions attending the formation of the boghead are thus summarized. Towards the end of the filling-up of the Permian lake of Autun, and when the rivers discharging into it had become mere torrents, an extraordinary vegetation of microscopical algæ, favoured by a period of calm, covered the whole surface, all other species of plants disappearing before it. Falling to the bottom they carried with them grains of dust and sand, remains of plants and other suspended matter, which latter, the "fundamental matter" above spoken of, forms 0·245 of the mass. On the bed becoming exposed and drying up, cracks were produced which were filled up by waters containing silica. It was also subsequently exposed to waters carrying carbonate of lime and magnesia.

Other beds of so-called "false boghead" exist above the workable bed, consisting of the same algæ, but intermixed with a larger proportion of the same "fundamental matter," and doubtless of precisely similar origin. But, furthermore, the whole thickness of the Permian formation at Autun—some 4,000 feet—contains more or less of the same fossil remains, although its bituminous character is not due entirely to those particular organisms. G. E. C.

SUPPOSED OCCURRENCE OF COAL IN THE EIFEL, WESTPHALIA.

Sur la présence prétendue de la houille dans l'Eifel. *By* G. DEWALQUE. *Annales de la Société Géologique de Belgique, (Bulletin),* 1893, *vol. xx., pages lxii.-lxiv.*

The statements that coal has been found in the Eifel are very positive, but the manner in which they are made shows that those making them are not well up in the geology of the district. The coal found has been variously described as a true coal, a cannel coal, and as a bituminous anthracite. A block of 66 lbs. weight is said to have been raised from the Elise mine at Gerolstein. At Neunkirchen, a bed 6½ feet thick has been found at a depth of from 42½ to 46 feet. It was found on trial to leave 15 to 20 per cent. of ash. This seam has been traced and mapped from Mürlenbach by Salm, Wallenborn, Oberstadfeld, Neroth, and Neunkirchen to Gefall and Utzerath.

The author of this note does not accept these sanguine views. In the first place, the strata are not Coal-measures but Devonian, and it is pointed out that in certain of the rocks associated with the coal-seam impressions of *sea weeds* were found. Abundance of vegetable *débris* has for long been known to occur in that part of the Devonian series where the seam occurs. It is concluded that the coal is merely an anthracite too impure to be worked with profit.

G. W. B.

CONTINUATION OF THE SAARBRÜCK COAL-MEASURES BENEATH YOUNGER ROCKS.

Setzt die Saarbrücken Steinkohlenformation unter dem pfälzischen Deckgebirge fort? *By* L. ROSENTHAL. *Zeitschrift für Praktische Geologie,* 1894, *pages* 88-91.

The productive Coal-measures occupy only a relatively small surface area in the extreme south-west of the Saar-Rhine region. The rest of the district is occupied by Upper Coal-measures and Lower Permian, along with eruptive rocks (melaphyre and porphyry). The area under which Coal-measures may occur under more recent formations is considerable, being about 62 by 19 miles.

Near the borders of this area a shaft has been sunk in connexion with the Frankenholz colliery, through the covering of Upper Coal-measures. At a depth of 504 feet, the upper flame coal group of the Saarbrück seams was reached.

About 2½ miles farther north-east another shaft has been sunk. Judging from the nature and succession of the rocks passed through, this shaft ought also to reach the Saarbrück coal-seams. Coal has been met with, but too much disturbed to be workable. In February, 1894, this shaft had reached a depth of 2,035 feet, and it is anticipated that it will soon reach the Holzer conglomerate, which forms the base of the *Leaia* stage of the Upper Coal-measures.

Farther north still, between Hocherberg and Potzberg, a boring has been made. Three coal-seams have been found at depths of 816, 826, and 855 feet respectively. These may correspond to the so-called "Hangenden Flotzzug" of the Saar district.

From these facts it may be assumed that the productive Coal-measures occur beneath the surface-covering of younger rocks; but in what thickness is yet unknown.

In the Saar district the thickness of the Coal-measures is unknown, and although a depth of 9,100 feet has been proved, the base has nowhere been reached. It has been said that the Saar strata thin out more and more in their north-easterly extension. Dr. Lappla, however, has shown that this is not so.*

Whether productive Coal-measures can be worked under the Permian cover over the whole of the Saar-Rhine district is more than doubtful. Depths of 3,250 and 4,000 feet are at present beyond the skill of the mining engineer, and therefore the greater part of the region in question must be regarded as practically unproductive. There remains, however, the districts around Höcherberg, Potzberg, Hermansberg, Königsberg, and Lemberg, where the miner may reasonably hope to win the coal through the unproductive covering. G. W. B.

THE AGE OF THE COAL-FIELD OF COMMENTRY, FRANCE.

Sur le Synchronisme des Bassins houillers de Commentry et de Saint-Etienne et sur les Conséquences qui en découlent. *By* M. A. JULIEN. *Comptes Rendus des Céances de l'Académie des Sciences,* 1894, *pages* 155-158.

The relative age of this coal-field must be determined by a comparison of its flora with that of Saint-Étienne. This latter has been recognized as the type of Upper Coal-measures since the publication of Mr. Grand 'Eury's valuable work. The neglect of this method has led to the prevailing misconceptions as to its age. As a preliminary step a table has been prepared showing the division of the flora of Saint-Étienne into nine stages. The flora of the Commentry coal-field contains 210 species. About 50 per cent. of these are common to the two basins. These have been divided into groups and compared with those of Saint-Étienne. By these means the author is able to synchronize the beds of the two basins. Thus the roof of the thick coal-seam (*Grande Couche*) and intercalated schists are equivalent to the Middle and Upper stages of the Saint-Étienne coal-field. G. W. B.

THE BHAGANWALA COAL-FIELD, INDIA.

Report on the Bhaganwala Coal-field, Salt Range, Punjab. *By* TOM D. LA TOUCHE. *Records of the Geological Survey of India,* 1894, *vol. xxvii., pages* 16-33, *map and two plates.*

This coal-field is situated on the plateau overlooking the village of Bhaganwala, near the eastern end of the Salt Range. Bhaganwala itself is about 10 miles north-east of Haranpur station, on the Sind-Sagar railway. The area forms a part of the Eastern plateau. The existence of coal here has been known since 1863, but it is only quite recently that an attempt has been made to open out the coal-field.

In 1864, Dr. Oldham estimated that the total quantity of available coal was between 50,000 and 60,000 tons.

In 1892, Mr. Luckstedt reported that the quantity of coal was estimated at 10,000,000 tons, of which 6,000,000 were said to be available. Before sanctioning the expense necessary to provide railway communication further investigation was

* *Zeit. f. Prak. Geol.*, 1893, page 393.

thought advisable, and the present survey was undertaken. While it was in progress Mr. Luckstedt furnished another report, in which the amount of coal was estimated at 20,000,000 tons.

The author's conclusion, taking into account only those areas where the existence of workable coal has been actually proved, is about 88,480 tons.

This is considered actually proved, while 1,000,000 tons is considered a reasonable estimate of the quantity obtainable. Thus it would be quite worth while to improve existing communications. G. W. B.

COAL IN HOLLAND.

Stein- und Braunkohlen in Holland. By B. *Zeitschrift für praktische Geologie*, 1894, *pages* 28-29.

Near Heerlen, in the province of Limburg, a boring proved the existence of flat regular strata of coal. The coals are identical with those of the Aix-la-Chapelle coal-basin, and the strata are the same but flatter and more regular. The coal was reached at 325 feet, and the boring passed through 78 feet of coal-bearing strata.

At other places, near Heerlen, beds of brown coal, over 22 feet in thickness, have been found. G. W. B.

THE BIG STONE GAP COAL-FIELD, VIRGINIA, U.S.A.

By J. M. Hodge. *American Institute of Mining Engineers*, 1893, *vol. xxi., pages* 922-938.

The Big Stone Gap coal-field embraces about 150 square miles, but its boundaries have not yet been accurately determined. It is situated in Virginia and Kentucky.

The Coal-measures are about 3,000 feet thick. The lower 1,000 feet or conglomerate series contains three to five beds of coal, but has not as yet been thoroughly explored. The upper two-thirds or lower productive measures contain fully twenty coal-seams. The characteristic rocks are the lower conglomerate and a persistent fossiliferous limestone, 1 to 3 feet thick, about 1,200 feet above it.

One section taken through the measures shows 50 to 60 feet of workable coal. The three most important seams are the high splint coal, 5 to 7 feet thick, of very good quality, especially for household purposes; next, a bed of bituminous coal, 4 to 6 feet thick, yielding an excellent coke; and lastly, the Imboden bed, varying from 5 to 20 feet thick, which yields an exceptionally good coke, low in ash and sulphur and high in fixed carbon. A. W. G.

REFUSE IN ANTHRACITE, U.S.A.

Quantity of Refuse in Anthracite Coal-beds. By Wm. Griffith. *Engineering and Mining Journal (New York)*, 1892, *vol. liii., page* 494.

The author had recently ascertained the approximate amount of refuse contained in the coal-seams of the Northern Pennsylvania anthracite field, for the four districts of Scranton, Pittston, Wilkes-Barre, and Panther Creek basin (between Mauchchunk and Tamaqua.

The author finds that the lowest beds contain the most refuse, while the higher, or newer, beds contain the least. This at once suggests the idea of a general law to this effect, in which event the bituminous, which are supposed to be of later origin than the anthracite coal-seams should also carry a smaller percentage of refuse.

J. W.

THE DISCOVERY OF COAL UNDER CREMORNE, SYDNEY HARBOUR.

By T. W. E. DAVID *and* E. F. PITTMAN. *Records of the Geological Survey of New South Wales*, 1894, *vol. iv., pages* 1-7, *with two plates.*

The existence of coal under Sydney has been regarded by geologists as almost a certainty, the only point upon which there was any divergence of opinion being the depth at which it was likely to be found.* Twelve borings have been made in or near Sydney with the object of proving the depth at which the coal will be found.

The Newington boring, on the Paramatta river, was made in 1878, and was abandoned at a depth of 1,312 feet without reaching coal. The chocolate shales were reached at a depth of 1,100 feet.

The Holt-Sutherland No. 1 (Botany) boring was abandoned in 1879 at a depth of 2,193 feet without reaching coal. The chocolate shales were entered at a depth of 880 feet.

The Moore Park boring, south of Sydney, was abandoned in 1880, at a depth of 1,860 feet without reaching coal. The chocolate shales were reached at a depth of 1,043 feet.

The Holt Sutherland No. 2 (Botany) boring was relinquished at a depth of 1,875 feet without reaching coal.

The Camp Creek boring, near the Metropolitan Coal Company's Helensburgh colliery, proved a coal-seam, 12 feet thick, at a depth of 846 feet. The chocolate shales are seen at the surface near this borehole.

The Heathcote boring is near the Waterfall station on the Illawarra Railway. The total depth bored was 1,586 feet, proving the upper seam 4 feet 8½ inches thick, at a depth of 1,513 feet, and the lower seam 6 feet 1 inch thick, at a depth of 1,578 feet. The chocolate shales were found at a depth of 307 feet.

The Holt-Sutherland No. 3 (Dent's Creek) boring was completed in 1887. The total depth bored was 2,307 feet. The chocolate shales were found at 787 feet, and two seams of coal: the upper seam 4 feet 2 inches thick, at a depth of 2,228 feet, and the lower seam 5 feet 3 inches thick, at 2,296½ feet.

The Narrabeen boring was abandoned at a depth of 1,985 feet without having reached coal. The chocolate shales were found at a depth of 379½ feet.

The Rose Bay (Sydney) boring was abandoned in 1888 without proving coal.

The Moorebank boring, near Liverpool, was made to a depth of 2,601½ feet, and proved three seams of coal: the upper one, 1 foot 5 inches thick, at a depth of 2,493½ feet; the second, 1 foot 4 inches thick, at a depth of 2,507½ feet; and the third, 6 feet 6¼ inches thick, at a depth of 2,583 feet.

The No. 1 Cremorne boring is situated north of Sydney Harbour. The boring was made to a depth of 3,095 feet, the chocolate shales being found at a depth of 943 feet; and a seam of coal, 7 feet 3½ inches thick, was found at a depth of 2,802 feet. This coal was charred, owing to the intrusion of two dolerite dykes, whose course was indicated on the surface.

The No. 2 Cremorne boring was made about ½ mile north-west from the No. 1 boring, and proved, at a depth of 2,917 feet a seam of coal 10 feet 3 inches thick. This seam, from analysis, appears superior to the Illawarra or southern coal. The temperature at the bottom of the boring was 97 degs. Fahr., equivalent to an average increase of 1 deg. Fahr. for every 80 feet of descent. M. W. B.

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* Dr. J. R. M. Robertson on "The Coal-fields South of Sydney, New South Wales," *Trans. Fed. Inst.*, vol. iv., page 83, and Plate VIII.

COBALT IN THE ASTURIAS, SPAIN.

Los Criaderos de cobre y cobalto del Aramo (Asturias). *By* ROMAN ORIOL. *Revista Minera*, 1893, *vol. xliv., pages* 390-392.

The Aramo deposits are in many respects similar to the cobaltiferous belt of Villamanin, Leon, and are found in the same geological horizon. The author considers the dolomite bands which contain the deposits in both cases to be merely altered beds of the limestone, the foldings of which they follow; but considers the mineralization of the dolomites in the latter case to have been contemporaneous, whereas in the former he supposes it to have been produced subsequently by lateral secretion. The Aramo deposits are therefore on this theory true "mantos" (beds), of which he recognizes at least three. G. E. C.

COPPER DEPOSITS OF LIMOGARDI, GREECE.

Étude sur le gisement cuivreux de Limogardi, montagnes de l'Othrys (Grèce). *By* G. A. GEORGIADES. *Bulletin de la Société de l'Industrie Minérale*, 1893, *series* 3, *vol. vii., pages* 143-153.

The crest of the Othrys chain runs east and west, separating the province of Phthiotis from Thessaly. The deposit in question is situated in the bottom of a ravine formed by two of its southern spurs, at an elevation of 2,500 feet, and at an equal distance from Lamia, the capital of Phthiotis, and the port of Stylida. It is connected with the latter by a new road, practicable for carts, about 10½ miles long.

There are extensive ancient workings and slag-heaps, which, judging from coins found and other indications, date back to about the third century B.C. These workings, following the strings of oxidized ores, are very irregular; there are, however, later workings, consisting of regular levels in good preservation, thought to be of Roman date.

The Othrys chain consists of clayey, talcose, and magnesian schists, overlain by Cretaceous limestone, and cut by diorite dykes, in the vicinity of which the schists are much altered and mineralized.

At the Limogardi mines, as elsewhere, the mineralization seems to be caused by a number of diorite dykes, at the outcrop of which, as also on the contact with the enclosing schists, veinlets and pockets of quartz, carrying the usual secondary copper ores, are found. The old workings, which in no case extend below water-level, were made only on these.

The main deposits, however, are impregnations of sulphide ores (chalcopyrite, mispickel, phillipsite, etc.) in the diorite, which from their lowness of grade, not exceeding 4½ per cent., and the hardness of the rock, were left untouched. These the author thinks might easily be enriched by dressing, on which treatment the future of the enterprise depends. G. E. C.

CUPRIFEROUS DEPOSITS OF VOLCANIC ORIGIN IN THE CAUCASUS.

Note sur un Gîte cuivreux d'Origine volcanique du Caucase méridional. *By* —. CHAPER. *Bulletin de la Société Géologique de France*, 1893, *series* 3, *vol. xxi., pages* 101-109.

Fifty miles, as the crow flies, south of Tiflis rises in a mountainous district the ancient volcano of Lelvar, whose activity began in the remote Jurassic age and lasted in all probability till near the close of Cretaceous times. This extinct crater is surrounded by a belt of metalliferous deposits, whereof the author only examined

those at three localities—Akhtala, Chamluk, and Allah Verdi, situated north-east of Mount Lelvar, at the respective heights above sea-level of 2,267, 3,887, and 3,835 feet. From the commercial point of view the deposits do not appear to have fulfilled the expectations which had been formed of them, although much money has been spent in their development.

The author considers that the origin of these deposits is precisely analogous to the origin of the El Boleo metalliferous ores in Lower California, and he traces as follows the history of the Caucasian deposits:—At the commencement of Jurassic time a ring of lakes surrounded the base of the Lelvar. At the bottom of these lakes thermal springs gushed forth, charged with silica and silicates, forming great masses of tufa about 100 feet thick. The tufa is now entirely made up of silica, the alkalies having all been leached out in the course of ages. But a new system of fractures opened in the bottom of the lake-basins, and sulphides of iron and copper were brought in by highly mineralized springs. At first these two metals alone were deposited, but later on others—such as lead, zinc, barium, and calcium came in. The ores accumulated in small lenses and kidney-shaped or mushroom-shaped masses above the vents by which they were brought in. All the sulphides are well crystallized, and are included unaltered in the siliceous matrix, from top to bottom of the deposit. That the whole area was subsequently depressed beneath the sea-level is proved by the fact that, at Chamluk, marine sedimentary deposits overlie directly the siliceous tufa. At Allah Verdi there intervene, however, andesite (basalt) and quartziferous andesitic tuff or breccia. At Akhtala a mass of metamorphosed lava, now nearly all quartz, with traces of epidote giving it a greenish hue, lies upon the tufa, and over this comes a great thickness (650 feet) of marine sedimentaries (interbedded with, and more or less altered by, lava-flows) betraying the continuance, during (and after?) their deposition, of volcanic activity. The fossils from the marine beds belong to the Middle Jurassic fauna. The author points out that the previously accredited, widely spread statements that all the rocks of the district are Cretaceous and post-Cretaceous limestones, etc., are erroneous; the difficulty at Akhtala is to get hold of any lime at all. O. S. E.

THE COPPER ORES OF PERSIA.

Die Kupfererze Persiens. *By* A. F. Stahl. *Chemiker Zeitung*, 1893, *vol. xviii., pages* 3-4.

Up to the last two hundred years the copper used in Persia was produced entirely in the country; the quantity, however, can never have been great. Only the richer oxidized ores were smelted, ores of 3 to 5 per cent. being now visible in the mines where left by the workers; and the old slags contain from 1 to 3 per cent. of copper. The present production is uncertain, but the rents paid to the Government show it to be inconsiderable. There is little prospect of any extensive works being established, as excepting near the Caspian, where, on the north slopes of the Elburz mountains, wood and coal are available for fuel, or near the Persian gulf, (in neither of which regions is there sufficient evidence of important deposits) the cost of carriage would be prohibitive.

The district of Anarek is rich in minerals. It is composed of recent limestones resting on ancient eruptive rocks and crystalline schists, which are broken through by trachytic masses of Tertiary age. Unfortunately water and fuel are scarce; copper and lead are, however, smelted with charcoal made from the bushes found in the desert. Mining, together with camel-breeding, has supported a population of some 2,500 people probably for centuries.

East of Semnan, some 10 miles from the petroleum-wells, carbonate and oxide of copper occur as an impregnation in clay-slate, the mineralized stone assaying 5 per cent. Should the borings here succeed, the deposits would certainly be worth working, as sheet copper would be needed for the transport of the oil.

The only copper deposit seen by the author which seemed to be payable was situated about 10 miles east of Dorfa Agere, at the foot of the Sefid-kuh mountains. Here there are two veins of copper pyrites, 3 to 4 inches wide, with fluor and calcspar, at the contact between porphyritic rock and Jurassic limestone. Coal is found in the vicinity, and communications both with the coast and the interior are comparatively good. It would be worth while to develop these veins in order to ascertain whether they are worth working.

In conclusion, the author says that though Persia appears to be rich in copper ores, the workable deposits are few and far between. G. E. C.

UNDERGROUND DAMS.

Über die im Steinkohlenwerke des Oberhohndorfer Schacht-Steinkohlenbau-vereins in den Jahren, 1891 *und* 1892, *zur Ausführung gelangten Wasserhaltungsbauten. By* V. NEUKIRCH. *Jahrbuch für das Berg- und Hüttenwesen im Königreiche Sachsen*, 1893, *pages* 108-136, *and five plates.*

The coal-seams were broken by a fault, and in order to keep back the water in the workings on one side of the fault, which were above the workings on the other, it was necessary to fill in and make watertight a shaft and three cross-cuts. The shaft was filled in with ashes and sand, and made watertight by having in places three feet of lime stamped down upon it. The lime was specially dried in the coke-ovens, then ground to powder, and afterwards dried again. This dam was found very successful in resisting water, and the author thinks he might have saved the cost and labour of a brick dam, which had been previously designed, and which, for the sake of double security, was built. The work cost £3,450, and has proved entirely successful. W. F. W.

NOTES ON DIAMONDS AND EMERALDS.

(1.) *Sur un Diamant à éclat d'Argent natif. By* ED. JANNETAZ. (2.) *Sur l'Existence du Diamant dans le Fer météorique de Cañon Diablo. By* CH. FRIEDEL. (3.) *Note sur le Calcaire Noir renfermant les Émeraudes de Muso (Nouvelle Grenade). By* ED. JANNETAZ. *Bulletin de la Société française de Minéralogie*, 1892, *vol. xv., pages* 131-133, 237, 258-263.

(1.) The Paris Museum of Natural History has lately been the recipient of an octahedric diamond of 27 carats, the faces of which are "shagreened" by innumerable little crystals within the diamond meeting the face-planes along certain lines. Each of these crystals reflecting practically all the light that it receives, assumes a metallic brilliancy, and at the first glance the diamond might be taken for a crystal of native silver.

(2.) A selected specimen of meteoric iron from Cañon Diablo (Arizona) having been submitted to the appropriate chemical treatment at the Paris School of Mines, has been proved to contain diamond of the *carbonado* variety, that is, small agglomerations of minute opaque grains, showing no crystalline form, but very lustrous and dark brownish-grey in colour. The author points out that the diamond, thus disseminated as a fine powder in portions of the meteorite, more

particularly in association with certain included nodules of phosphide and sulphide of iron, appears to have been formed *in situ* during crystallization or consolidation of the mass. This is said to be the first occasion on which diamond has with certainty been found in meteoric iron (the analogous cases of the meteorite of Novo-Urei, Penza, in Russia, and of the Arva iron have been hitherto looked upon with some suspicion) which the author regards as the true original matrix of the precious mineral.

(3.) Mr. Jannetaz points out that the black dolomitic limestone, which is the matrix of the emeralds of Muso (New Granada) owes its colour to the presence of anthracite. He has been able to isolate fragments of anthracite from among the dolomite-crystals amid which they are entangled. It appears probable, despite Wœhler's statement that the splendid green colouration of the Muso emeralds arises from the presence of [? compounds of] chromium, that some of the darker gems do contain a little carbon disseminated within them in the form of minute particles. But this apparently would not apply to the dark mica-schist which is the matrix of the well-known Egyptian emeralds. Some of the blackest mica-schist from Jebel Zabarah has been tested for carbon, and found to contain none. O. S. E.

NOTE ON BOILER EXPLOSIONS.

By Prof. Wm. P. Mason. *American Institute of Mining Engineers*, 1883, *vol. xxi., pages* 374-277.

The paper gives the results of experiments made with a view to contradict the statement that "closed metallic vessels, partly filled with water and heated, do not become shattered by violent explosion, but simply rupture by over-pressure."

The author, after experimenting with several solid brass tubes, $1\frac{1}{2}$ inches in diameter, by half-filling with water and then heating, explains that a rupture would be caused by gradually increased pressure, and an explosion would result from the rupture on account of the instantaneous and material reduction of such pressure, leaving the water violently super-heated.

Furthermore, that it is possible to produce a rupture too small in area to reduce the pressure in a boiler to atmospheric pressure with sufficient suddenness to cause the whole body of water to spring into steam, and thus a "blow off" would be instituted in place of an explosion. A. W. G.

THE HILL-FARM-PARRISH MINE-FIRE.

By F. A. Hill. *American Institute of Mining Engineers*, 1893, *vol. xxi., pages* 632-643, *and plate.*

The Hill-Farm-Parrish mine is situated in the heart of the Connellesville coke region, Pennsylvania.

The fire was caused by igniting a feeder of gas from a borehole which set fire to a cloth brattice, and quickly spread up one of the timbered slopes until it was all on fire. There was no explosion and no after-damp. Wood smoke filled the mine and killed 31 workmen. After several unsuccessful attempts the mine was reopened in April, 1891, ten months after the disaster.

The writer shows that the coking action of the fire on the face of the coal prevented much loss from burning by shutting off the air, that the fire in the timber was only overcome with great difficulty, and that the bituminous slate of the district sustains fire better than the coal. A. W. G

THE NORTH OF ENGLAND INSTITUTE

OF

Mining and Mechanical Engineers.

ANNUAL REPORT OF THE COUNCIL

AND

ACCOUNTS FOR THE YEAR 1893-94;

LIST OF

COUNCIL, OFFICERS, AND MEMBERS,

FOR THE YEAR 1894-95;

AND

THE CHARTER AND BYE-LAWS.

1893-94.

NEWCASTLE-UPON-TYNE: PUBLISHED BY THE INSTITUTE.

PRINTED BY ANDREW REID & CO., LTD., LONDON AND NEWCASTLE-UPON-TYNE.

1894.

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THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

ANNUAL REPORT OF THE COUNCIL.

The continued increase in the number of the membership during recent years is shown in the following table:—

	August 1st.		
	1892.	1893.	1894.
Honorary Members...	24	25	26
Members	545	565	624
Associate Members	35	47	53
Associates	34	47	68
Students	35	34	20
Subscribing Collieries	18	22	22
Totals	691	740	813

One hundred and nineteen members of all classes have joined the Institute during the year, and after allowing for the losses, through death, resignations, and non-payment of subscription, the net increase is 73. The members may be congratulated upon the increasing prosperity of the Institute, and the Council desire them, at all times, to promote its claims and advantages, and to extend the membership.

The Explosives Committee have carried on investigations at the experimental station, and they anticipate being able to issue the report upon "flameless explosives" in presence of gas, before the end of the year. The appliances and apparatus were inspected by several of the members of the Royal Commission on Explosions from Coal-dust in Mines, who expressed great satisfaction with the experiments and results obtained by the committee.

The "Report of the Prussian Fire-damp Commission," translated by Dr. P. P. Bedson and Mr. L. L. Belinfante; and "Mining Explosives: Their Definition as Authorized under the Explosives Act (1875)," by Mr. A. C. Kayll, have been of great use to the Committee. The papers on "A Contribution to our Knowledge of Coal-dust," by Dr. P. P. Bedson and Mr. W. McConnell, and on "The Combustion of Oxygen and Coal-dust in Mines," by Mr. W. C. Blackett, will also be of value when

explosives are tested in presence of coal-dust. Mr. J. D. Kendall's paper on "Miss-fires" confirms some of the results obtained and the practice of the Committee in their experiments.

The library has been maintained in an efficient condition during the year. The additions by donations, exchange, and purchase have been :—

Bound volumes	265
Pamphlets, reports, etc.	245
A total of	510 titles.

And there are now at least 6,666 volumes and 1,267 unbound pamphlets, etc., in the library. Owing to this large increase, bookshelves have been placed in the corridor adjacent to the lecture-theatre, and ample space has been thus obtained for further additions to the library.

The Wood Memorial Hall, lecture theatre, etc., are now lighted by electricity, which has materially added to the convenience of these rooms.

The Watson collection of mining-plans and papers has been placed in the corridor, where they may be examined, but not removed, by members.

The reservation of a reading-room has been under consideration, and the Council request the views of members upon the question.

The fifth volume (S—T) of the "Sinkings and Borings" has been completed, and may be purchased by the members. The Council trust that the remaining volumes will be issued at early dates.

Members who desire to complete their sets of the *Transactions* and other publications of the Institute may now do so at the recently revised prices.

The files of some of the *Transactions* of societies in the library are incomplete owing to loss of volumes or parts, and it is desired that members will return the missing volumes.*

The value of the library would be materially increased if members and friends would kindly present books, maps, etc., which can be spared from their own libraries.

The Federated Institution of Mining Engineers has now been in existence for five years, and during the last year general meetings have been held at Glasgow on September 5th, 6th, and 7th, 1893; at Leeds on February 14th, 15th, and 16th, 1894; and at London on June 7th, 8th, and 9th, 1894. The meetings have been well attended, and the status of the Institution enhanced by the papers communicated and the accompanying discussions. With continued energy on the part of the

* See list accompanying the Annual Report of the Council for 1892.

Federated Institutes, The Federated Institution of Mining Engineers should include the whole of the societies interested in mining and metallurgical engineering.

The report of the committee on mechanical ventilators is approaching completion, and will, it is hoped, be issued during the current year to the members.

The Council have requested Prof. J. H. Merivale to represent the Institute at the conference of delegates of corresponding societies at the ensuing meeting of the British Association for the Advancement of Science at Oxford.

Prizes of books have been awarded to the writers of the following papers, communicated to the Institute during the year 1892-93 :—

"Use of Cement in Shaft-sinking." By Mr. Bennett H. Brough.

"Observations on Fans of Different Types Working on the same Upcast Shaft." By the Rev. G. M. Capell.

"Manometric Efficiency of Fans." By the Rev. G. M. Capell.

"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment." By Mr. A. G. Charleton.

"Steam Boilers with Forced Blast: the Perrett System for Burning Dust and Rejected Fuels; with Notes on Testing Boilers." By Mr. Bryan Donkin, Jun.

"The Gold-bearing Veins of the Organos District, Tolima, U.S. Colombia." By Mr. Edward Halse.

"The Education of Mining Engineers." By Prof. J. H. Merivale.

"The Geology and Coal-deposits of Natal." By Mr. R. A. S. Redmayne.

"Gold-mining in Brazil." By Mr. E. M. Touzeau.

The papers communicated to the Institute have been :—

"A Contribution to our Knowledge of Coal-dust, Part II." By Dr. P. Phillips Bedson, D.Sc. (Lond.).

"A Contribution to our Knowledge of Coal-dust, Part III." By Dr. P. Phillips Bedson, D.Sc. (Lond.), and Mr. W. McConnell, Jun.

"The Combustion of Oxygen and Coal-dust in Mines." By Mr. W. C. Blackett.

"History and Description of the Greenside Silver-lead Mine, Patterdale." By Mr. W. H. Borlase.

"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment." By Mr. A. G. Charleton.

"The Ghorband Lead-mines, Afghanistan." By Mr. A. L. Collins.

"Note on the Antimony Deposit of El Altar, Sonora, Mexico." By Mr. Edward Halse.

"Note on the Occurrence of Mercury at Quindiú, Tolima, U.S. Colombia." By Mr. Edward Halse.

"Description of the Whitehaven Collieries." By Mr. H. M. James.

"Mining Explosives: Their Definition as Authorized under the Explosives Act (1875)." By Mr. A. C. Kayll.

"Miss-fires." By Mr. J. D. Kendall.

"A Short Description of the Hæmatite Deposit worked by the Salter, Eskett, and Winder Gill Mines, and the Method of Working It." By Mr. J. D. Kendall.

"Singareni Coal-field, Hyderabad, India." By Mr. J. P. Kirkup.

"Corliss-engined Fan at Seghill Colliery." By Mr. C. C. Leach.

"Historical Sketch of the Whitehaven Collieries." By Mr. R. W. Moore.

"Description of the St. Helens Colliery, Workington." By Mr. G. Scoular.

"Presidential Address." By Mr. A. L. Steavenson.

"Magnetic Declination and its Variations." By Prof. H. Stroud.

"Minerals and Mining in Tasmania." By Mr. A. P. Wilson.

The Council congratulate the members on the success of the general meeting recently held in Cumberland and Westmorland, and consider that the best thanks of the Institute are due to the committee who made the arrangements. The Council also direct the attention of the members to the great courtesy shown by the Whitehaven Collieries Company, the Postlethwaites Eskett Mining Company, the Cleator Iron-ore Company, the Winder Gill Iron-ore Company, the St. Helens Colliery and Brickworks Company, and the Greenside Mining Company, and have great pleasure in recording the thanks of the Institute to all who by their services contributed to the holding of that successful meeting.

FINANCE REPORT.

The ordinary income for the year 1893-94 amounted to £1,898 4s. 7d., being an increase over the previous year of £86 10s. 4d.

The total receipts from subscriptions and arrears, excluding amounts paid in advance, amounted to £1,567 9s., an increase of £106 18s. over the preceding year.

The amount of subscriptions now in arrear, after striking off those which are considered irrecoverable and excluding the current year, is £29 8s.

The total expenditure amounted to £1,909 16s. 8d., an increase of £177 1s. 2d. The exceptional items contributing to this increase are: Contributions to The Federated Institution of Mining Engineers, owing to the increased number of members, £55 14s. 5d.; electric light fittings in the Wood Memorial Hall and Lecture Theatre, £57; book-shelves in the corridor adjoining the Lecture Theatre, £87 4s.; and an increase on account of postages of £12 0s. 3d., due to the increased number of members. There has, however, been a decrease of £29 1s. 4d. in the expenditure on books for the library, and of £18 18s. on the amount awarded for prizes, owing to last year's balance sheet, including the prizes awarded for two years.

DR. THE TREASURER IN ACCOUNT WITH THE NORTH OF ENGLAND

FOR THE YEAR ENDING

	£ s. d.	£ s. d.
July 22nd, 1893.		
To Balance at Bankers	468 3 8	
„ „ in Treasurer's hands	84 12 3	
„ Outstanding Amounts for Authors' Excerpts	11 15 4	
		564 11 3
„ Dividend of 7½ per cent. on 134 Shares of £20 each in the Institute and Coal Trade Chambers Co., Ltd., for the year ending June, 1894	201 0 0	
„ Interest on Investments with the River Tyne Commissioners	53 8 0	
		254 8 0
To SUBSCRIPTIONS FOR 1893-94 AS FOLLOWS:—		
466 Members ... ... @ £2 2s.	978 12 0	
33 Associate Members ... ... @ £2 2s.	69 6 0	
41 Associates ... ... @ £1 1s.	43 1 0	
26 Students ... ... @ £1 1s.	27 6 0	
87 New Members ... ... @ £2 2s.	182 14 0	
1 New Member	25 0 0	
10 New Associate Members ... ... @ £2 2s.	21 0 0	
15 New Associates ... ... @ £1 1s.	15 15 0	
5 New Students ... ... @ £1 1s.	5 5 0	
	1,367 19 0	
To SUBSCRIBING COLLIERIES, VIZ.:—		
Ashington Coal Company £2 2 0		
Birtley Iron Company 6 6 0		
Bridgewater Trustees 6 6 0		
Marquis of Bute 2 2 0		
Butterknowle Colliery Company ... 2 2 0		
Cowpen Coal Company 4 4 0		
Earl of Durham 10 10 0		
Elswick Coal Company 2 2 0		
Haswell Coal Company 4 4 0		
Hetton Coal Company 10 10 0		
Hutton Henry Coal Company 2 2 0		
Marquess of Londonderry 10 10 0		
North Brancepeth Coal Company ... 2 2 0		
North Hetton Coal Company 6 6 0		
Ryhope Coal Company 4 4 0		
Seghill Coal Company 2 2 0		
South Hetton Coal Company 4 4 0		
Stella Coal Company 2 2 0		
Throckley Coal Company 2 2 0		
Victoria Garesfield 2 2 0		
Wearmouth Coal Company 4 4 0		
Westport Coal Company 4 4 0		
	96 12 0	
To Donations	4 4 0	
	1,468 15 0	
Less—Subscriptions for current year paid in advance last year	57 15 0	
	1,411 0 0	
To Arrears received	156 9 0	
	1,567 9 0	
„ Subscriptions paid in advance during current year ...	56 14 0	
		1,624 3 0
„ Sale of Publications		19 13 7
		£2,462 15 10

INSTITUTE OF MINING AND MECHANICAL ENGINEERS. CR.
JULY 31ST, 1894.

July 21st, 1894.	£	s.	d.	£	s.	d.
By Printing and Stationery				160	2	7
„ Books for Library	44	19	11			
„ Prizes for Papers	17	17	0			
„ Incidental Expenses	54	1	10			
„ Postages	57	5	5			
„ Sundry Accounts	16	17	6			
„ Travelling Expenses	7	6	1			
„ Salaries	150	0	0			
„ Clerks' Wages	152	7	4			
„ Reporter's Salary	12	12	0			
„ Rent	93	7	2			
„ Rates and Taxes	22	2	10			
„ Insurance	12	8	4			
„ Furnishing, Repairs, etc.	107	19	3			
„ Coals, Gas, Electric Light, and Water	21	3	1			
„ Electric Light and Fittings, etc.	57	0	0			
				827	7	9
„ Fan Committee	1	14	9			
„ Explosives Committee	81	16	1			
„ British Association Meeting—Delegate's Expenses ...	6	17	0			
				90	7	10
By Federated Institution of Mining Engineers—Subscriptions	847	10	3			
Less—Amounts paid by Authors for Excerpts ...	15	11	9			
				831	18	6
				1,909	16	8
By Balance at Bank	457	10	0			
„ „ in Treasurer's hands	84	12	3			
„ Outstanding Amounts for Authors' Copies	10	16	11			
				552	19	2
				£2.462	15	10

I have examined the above account with the Books and Vouchers relating thereto, and certify that, in my opinion, it is correct.

JOHN G. BENSON,
CHARTERED ACCOUNTANT.

Newcastle-upon-Tyne,
August 3rd, 1894.

DR. THE TREASURER IN ACCOUNT

			£	s.	d.	£	s.	d.
To 565 Members, 24 of whom are Life Members.								
541	541	@ £2 2s.				1,136	2	0
To 47 Associate Members, 5 of whom are Life Members.								
42	42	@ £2 2s.				88	4	0
To 47 Associates		@ £1 1s.				49	7	0
To 34 Students, 1 transferred to list of Members.								
33	33	@ £1 1s.				34	13	0
To 22 Subscribing Collieries		... ...				96	12	0
To 87 New Members		@ £2 2s.				182	14	0
To 1 New Member		@ £25				25	0	0
To 10 New Associate Members		@ £2 2s.				21	0	0
To 15 New Associates		@ £1 1s.				15	15	0
To 5 New Students		@ £1 1s.				5	5	0
To Donations						4	4	0
						1,658	16	0
To Arrears, as per Balance Sheet 1892-93			220	10	0			
Add—Arrears considered irrecoverable, but since received			10	10	0			
			231	0	0			
Less—Struck off as irrecoverable. Arrears		£45 3 0						
" " " Current year		27 6 0						
			72	9	0			
						158	11	0
						1,817	7	0
To Subscriptions Paid in Advance						56	14	0
						£1,874	1	0

WITH SUBSCRIPTIONS, 1893-94. CR.

		PAID. £ s. d.	UNPAID. £ s. d.
By 466 Members, paid	@ £2 2s.	978 12 0	.....
By 74 „ unpaid	@ £2 2s.	.....	155 8 0
By 1 „ dead	@ £2 2s.	.....	2 2 0
541			
By 33 Associate Members, paid	@ £2 2s.	69 6 0	.....
By 8 „ „ unpaid	@ £2 2s.	.....	16 16 0
By 1 „ „ dead	@ £2 2s.	.....	2 2 0
42			
By 41 Associates, paid	@ £1 1s.	43 1 0	.....
By 6 „ unpaid	@ £1 1s.	.....	6 6 0
47			
By 26 Students, paid	@ £1 1s.	27 6 0	.....
By 7 „ unpaid	@ £1 1s.	.....	7 7 0
33			
By 22 Subscribing Collieries, paid		96 12 0	.....
By 87 New Members, paid	@ £2 2s.	182 14 0	.....
By 1 New Member, paid	@ £25	25 0 0	.....
By 10 New Associate Members, paid	@ £2 2s.	21 0 0	.....
By 15 New Associates, paid	@ £1 1s.	15 15 0	.....
By 5 New Students, paid	@ £1 1s.	5 5 0	.....
By Donations		4 4 0	.....
		1,468 15 0	190 1 0
Less struck off as irrecoverable		.....	27 6 0
			162 15 0
By Arrears, paid		156 9 0	.....
„ „ unpaid		.....	29 8 0
By Subscriptions paid in advance		56 14 0	.....
		1,681 18 0	192 3 0
			1,681 18 0
			£1,874 1 0

Liabilities.	£	s.	d.	£	s.	d.
Subscriptions paid in Advance during the year	56	14	0			
„ carried over from previous year	2	2	0			
				58	16	0
Capital				9,380	7	8
				£9,439	3	8

I have examined the above account with the books and vouchers relating thereto, and certify that, in my opinion, it is correct. The Share Certificates and Bonds have been produced to me.

JOHN G. BENSON,
Chartered Accountant.

Newcastle-upon-Tyne,
August 3rd, 1894.

Assets.	£	s.	d.	£	s.	d.
Balance of Account at Bankers	457	10	0			
„ in Treasurer's hands	84	12	3			
Outstanding amounts for Authors' Excerpts ...	10	16	11			
				552	19	2
134 Shares of £20 each in the Institute and Coal Trade Chambers Co., Ltd.	.....			2,680	0	0
Invested with River Tyne Commissioners ...	.....			1,500	0	0
Arrears of Subscriptions	.....			192	3	0
Value of 541 Bound Volumes of Transactions, @ 8s. 6d.	229	18	6			
„ 4,398 do., @ 6s....	1,319	8	0			
„ 2,565 Unbound Parts, @ 1s.	128	5	0			
„ 77 Copies of Mr. T. F. Brown's Map, @ 2s. 6d.	9	12	6			
„ 372 Copies of Index, Vols. 1-25, @ 1s.	18	12	0			
„ 848 Copies of Catalogue of Fossil, Plants, @ 2s. 6d.	106	0	0			
„ 752 Copies of Fossil Illustrations, @ 5s.	188	0	0			
„ 1,500 Copies of Borings and Sinkings (Vol. I. in Sheets) @ 1s.	75	0	0			
„ 270 Do., Vol. I., @ 2s. 6d.... ...	33	15	0			
„ 301 Do., Vol. II., @ 2s. 6d.... ...	37	12	6			
„ 314 Do., Vol. III., @ 2s. 6d.... ...	39	5	0			
„ 420 Do., Vol. IV., @ 2s. 6d.	52	10	0			
„ 370 Do., Vol. V., @ 2s. 6d.... ...	46	5	0			
„ 254 Copies of Library Catalogue, @ 1s.	12	14	0			
„ 308 Copies of French Commission Report on Explosives in parts, @ 1s. ...	15	8	0			
„ 9 Do., Bound, @ 4s.	1	16	0			
				2,314	1	6
„ Office Furniture and Fittings ...	450	0	0			
„ Books and Maps in Library	1,750	0	0			
				2,200	0	0
				£9,439	3	8

OFFICERS, 1894-95.

PRESIDENT.

T. DOUGLAS, Esq., The Garth, Darlington.

VICE-PRESIDENTS.

W. ARMSTRONG, Esq., Pelaw House, Chester-le-Street.
T. W. BENSON, Esq., 11, Newgate Street, Newcastle-upon-Tyne.
W. F. HALL, Esq., Haswell Colliery, Haswell, *via* Sunderland.
J. L. HEDLEY, Esq., 22, Hawthorn Terrace, Newcastle-upon-Tyne.
G. MAY, Esq., Harton Colliery, near South Shields.
J. G. WEEKS, Esq., Bedlington, R.S.O., Northumberland

COUNCIL.

W. ARMSTRONG, Jun., Esq., Wingate, Co. Durham.
T. W. ASQUITH, Esq., Harperley Hall, Tantobie, R.S.O., Co. Durham.
W. C. BLACKETT, Esq., Acorn Close, Sacriston, Durham.
T. FORSTER BROWN, Esq., Guildhall Chambers, Cardiff.
T. E. FORSTER, Esq., 3, Eldon Square, Newcastle-upon-Tyne.
W. H. HEDLEY, Esq., Medomsley, R.S.O., Co. Durham.
T. HEPPELL, Esq., Leafield House, Chester-le-Street.
H. LAWRENCE, Esq., Grange Iron Works, Durham.
C. C. LEACH, Esq., Seghill Colliery, Northumberland.
Sir W. T. LEWIS, Mardy, Aberdare.
Prof. J. H. MERIVALE, 2, Victoria Villas, Newcastle-upon-Tyne.
H. PALMER, Esq., East Howle Colliery, near Ferryhill.
A. M. POTTER, Esq., Aydon Grange, Corbridge-upon-Tyne.
H. RICHARDSON, Esq., Backworth Colliery, Newcastle upon-Tyne.
R. ROBINSON, Esq., Howlish Hall, near Bishop Auckland.
T. O. ROBSON, Esq., Chowdene Cottage, Gateshead-upon-Tyne.
S. TATE Esq., Trimdon Grange Colliery, Co. Durham.
W. O. WOOD, Esq., South Hetton, Sunderland.

Ex-Officio. — Past-Presidents.

LORD ARMSTRONG, C.B., LL.D., D.C.L., F.R.S., Cragside, Rothbury.
LINDSAY WOOD, Esq., The Hermitage, Chester-le-Street.
G. B. FORSTER, Esq., 3, Eldon Square, Newcastle-upon-Tyne.
JOHN DAGLISH, Esq., Rothley Lake, Cambo, R.S.O., Northumberland.
SIR LOWTHIAN BELL, Bart., D.C.L., F.R.S., Rounton Grange, Northallerton.
WM. COCHRANE, Esq., St. John's Chambers, Grainger Street West, Newcastle-upon-Tyne.
J. B. SIMPSON, Esq., Hedgefield House, Blaydon-upon-Tyne.
A. L. STEAVENSON, Esq., Durham.
C. BERKLEY, Esq., Marley Hill, Swalwell, R.S.O., Co. Durham. — Retiring Vice-President.

TREASURER.

REGINALD GUTHRIE, Esq., Neville Hall, Newcastle-upon-Tyne

SECRETARY.

M. WALTON BROWN, Esq., Neville Hall, Newcastle-upon-Tyne.

LIST OF MEMBERS,

JULY 31, 1894.

PATRONS.

His Grace the DUKE OF NORTHUMBERLAND.
The Most Noble the MARQUESS OF LONDONDERRY.
The Right Honourable the EARL OF LONSDALE.
The Right Honourable the EARL OF DURHAM.
The Right Honourable the EARL GREY.
The Right Honourable the EARL OF RAVENSWORTH.
The Right Honourable the EARL OF WHARNCLIFFE.
The Right Reverend the LORD BISHOP OF DURHAM.
The Very Reverend the DEAN AND CHAPTER OF DURHAM.
WENTWORTH B. BEAUMONT, Esq.
BARON BARNARD.

HONORARY MEMBERS.

* Honorary Members during term of office only.

	Date of Election.
1 The Right Honourable the EARL of RAVENSWORTH, Ravensworth Castle, Gateshead-upon-Tyne	Nov. 3, 1877
2*J. B. ATKINSON, Esq., H.M. Inspector of Mines, Glasgow ...	Aug. 4, 1888
3*W. N. ATKINSON, Esq., H.M. Inspector of Mines, Newcastle, Staffordshire	Aug. 4, 1888
4*W. BEATTIE-SCOTT, Esq., H.M. Inspector of Mines, Great Barr, near Birmingham	Dec. 10, 1887
5*Prof. P. PHILLIPS BEDSON, Durham College of Science, Newcastle-upon-Tyne	Feb. 10, 1883
6*THOMAS BELL, Esq., H.M. Inspector of Mines, Durham ...	Aug. 4, 1888
7*Prof. G. S. BRADY, Durham College of Science, Newcastle-upon-Tyne	Nov. 6, 1875
8 Dr. BRASSERT. Berghauptmann, Bonn-am-Rhein, Prussia ...	Dec. 8, 1883
9 JOSEPH DICKINSON, Esq., South Bank, Pendleton, Manchester	Dec. 13, 1852
10*C. LE NEVE FOSTER, Esq.. H.M Inspector of Mines, Llandudno	Aug. 4, 1888
11*Prof. WILLIAM GARNETT	April 11, 1885
12*JOHN GERRARD, Esq., H.M. Inspector of Mines, Worsley, Manchester	June 11, 1892
13*HENRY HALL, Esq., H.M. Inspector of Mines, Rainhill, Prescott	March 4, 1876
14*J. L. HEDLEY, Esq., H.M. Inspector of Mines, 22, Hawthorn Terrace, Newcastle-upon-Tyne	April 9, 1892
15*Prof. A. S. HERSCHEL Observatory House, Slough, Bucks ..	Aug. 3, 1872
16 The Very Rev. Dr. LAKE, Dean of Durham	Feb. 3, 1872
17*Prof. G. A. LEBOUR, Durham College of Science, Newcastle-upon-Tyne	Nov. 1, 1879
18 J. A. LONGRIDGE, Esq , Grêve d'Ayetté. Jersey	Dec. 11, 1886
19*J. S. MARTIN, Esq , H.M. Inspector of Mines, Clifton	Aug. 4, 1888

Date of Election.

20 Prof. JOHN HERMAN MERIVALE, Durham College of Science, Newcastle-upon-Tyne ... ... Aug. 5, 1893
21* JOSEPH T. ROBSON, Esq., H.M. Inspector of Mines, Swansea Aug. 4, 1888
22*J. M. RONALDSON, Esq., H.M. Inspector of Mines, 44, Athole Gardens, Glasgow ... ... Aug. 4, 1888
23*A. H. STOKES, Esq., H.M. Inspector of Mines, Greenhill, Derby Aug. 4, 1888
24*Prof. H. STROUD, Durham College of Science, Newcastle-upon-Tyne ... ... Nov. 5, 1892
25 M. E VUILLMEIN, Mines d'Aniche, Nord, France ... ... Sept. 7, 1878
26*FRANK N. WARDELL, Esq., H.M. Inspector of Mines, Wath-on-Dearne, near Rotherham ... ... Feb. 4, 1865

MEMBERS.

Marked * have paid life composition.

Date of Election and of Transfer.

1 Aburrow, Charles, P.O Box 534, Johannesburg, Transvaal Feb. 13, 1892
2 Ackroyd, Wm., Morley Main Collieries, Morley, near Leeds Feb. 7, 1880
3 Adamson, Thos., E.I.R. Collieries, Giridih, Bengal, India Feb. 10, 1894
4 Agabeg, Frank J., Apcar and Co.'s Collieries, Sitarampore, India ... ... Dec. 9, 1893
5 Aitkin, Henry, Falkirk, N.B.... ... ... Mar. 2, 1865
6 Allan, John F, c/o Caldwell and Watson, 109, Fen- A.M. Feb. 10, 1883
church Street, E.C. ... ... M. June 8, 1889
7 Allan, T. Alexander, c/o Messrs. Gibbs, Bright, and Co., Melbourne, Australia ... ... Feb. 11, 1893
8 Allison, J. J. C., Woodland Collieries, Butterknowle, A.M. Feb 13, 1886
R.S.O., Co. Durham ... ... M. June 8, 1889
9 Anderson, C. W., Cleadon Park, Sunderland ... ... Aug. 21, 1852
10 Anderson, R. S., Elswick Colliery, Newcastle-upon-Tyne... S. June 9, 1883
A.M. Aug. 4, 1888
M. Aug. 3, 1889
11 Andrews, Hugh, Swarland Hall, Felton, Northumberland Oct. 5, 1872
12 Angus, James, Radcliffe, Acklington, Northumberland ... Oct. 8, 1892
13 Appleby, W. R., 911, Fifth Street, S.E., Minneapolis, Minn., U.S.A. ... ... Apr. 14, 1894
14 Archer, T., 6, Park Terrace, Gateshead-upon-Tyne ... July 2, 1872
15 Armstrong, Lord, C.B., LL.D., D.C.L., Cragside, Rothbury (Past-President, *Member of Council*) ... ... May 3, 1866
16 Armstrong, Henry, Chester-le-Street .. .. A.M. April 14, 1883
M. June 8, 1889
17 Armstrong, Wm., Pelaw House, Chester-le-Street (Vice-President, *Member of Council*) ... . Aug. 21, 1852
18 Armstrong, W., Jun., Wingate, Co. Durham (*Member of Council*) ... ... S. April 7, 1867
M. Aug. 6, 1870
19 Arnold, T., Castle Hill, Greenfields, Llanelly ... ... A.M. Oct. 2, 1880
M. June 8, 1889
20 Asquith, T. W., Harperley Hall, Tantobie, R.S.O., Co. Durham (*Member of Council*) ... ... Feb. 2, 1867
21 Atkinson, Claude W., The Electrical Coal Cutting Contract Corporation, Limited, 2, St. Nicholas' Buildings, Newcastle-upon-Tyne ... ... Aug. 6, 1892

		Date of Election and of Transfer.
22	Atkinson, Fred. R., West Street, Congleton	S. Feb. 14, 1874
		A.M. Aug. 4, 1883
		M. June 8, 1889
23	Atkinson, L. B., Messrs. T. W. Goolden and Co., Woodfield Works, Harrow Road, London	Aug. 6, 1892
24	Aubrey, R. C., The Midland Coal, Coke and Iron Co., Limited, Halmerend, near Newcastle, Staffordshire ...	Feb. 5, 1870
25	Austine, John, Cadzow Coal Co., Glasgow	Nov. 4, 1876
26	Ayton, Ernest F., El Bote Mining Negociacion, Zacatecas, Republic of Mexico	Aug. 1, 1891
27	Ayton, Henry, 122, Rye Hill, Newcastle-upon-Tyne ...	S. Mar. 6, 1875
		A.M. Aug. 2, 1884
		M. June 8, 1889
28	Bailes, E. T., Wingate, Ferryhill	A.M. June 7, 1879
		M. June 8, 1889
29	Bailes, T., Jesmond Gardens, Newcastle-upon-Tyne ...	Oct. 7, 1858
30	Bailey, Samuel, 30, Waterloo Street, Birmingham ...	June 2, 1859
31	Bain, R. Donald, 85, Pembroke Road, Clifton, Bristol ...	S. March 1, 1873
		M. Aug. 5, 1876
32	Bainbridge, E., Nunnery Colliery Offices, Sheffield ...	S. Dec. 3, 1863
		M. Aug. 1, 1868
33	Banks, Thomas, 60, King Street, Manchester	Aug. 4, 1877
34	Barker, M. W., P.O. Box 1463, Johannesburg, Transvaal	Apr. 8, 1893
35	Barrett, C. R., Whitehill Hall, Chester-le-Street	S. Nov. 7, 1874
		A.M. Aug. 7, 1880
		M. Dec. 11, 1886
36	Bartholomew, C., Castle Hill House, Ealing, London, W.	Aug. 5, 1853
37*	Bartholomew, C. W., Blakesley Hall, near Towcester ...	Dec. 4, 1875
38	Batey, John Newbury Collieries, Coleford, Bath	Dec. 5, 1868
39	Baumgartner, W. O., The Poplars, Cleadon Village ...	S. Sept. 6, 1879
		M. Aug. 3, 1889
40	Bayldon, Daniel Hy., 3, Drapers' Gardens, London, E.C.; letters to General Manager, New Clewer Estate G.M. Co., Limited, Pilgrim's Rest, Lydenburg, S.A.R. ...	Feb. 8, 1890
41	Beanlands, Arthur, Palace Green, Durham	Mar. 7, 1867
42	Bell, Sir Lowthian, Bart., Rounton Grange, Northallerton (Past-President, *Member of Council*)	July 6, 1854
43	Bell, C. E., Park House, Durham	S. Dec. 3, 1870
		M. Oct. 4, 1879
44	Bell, Geo. F., H. M. Inspector of Mines, 30, Eldon Street, Newcastle-upon-Tyne	S. Sept. 6, 1879
		A.M. Aug. 6, 1887
		M. Aug. 3, 1889
45	Bell, Walter, Ashington Colliery, Morpeth	S. Oct. 8, 1889
		M. Feb. 10, 1894
46	Bennett, Alfred H., Dean Lane Collieries, Bedminster, Bristol	A.M. April 10, 1886
		M. June 8, 1889
47	Benson, J. G., 12, Grey Street, Newcastle-upon-Tyne ...	Nov. 7, 1874
48	Benson, T. W., 11, Newgate Street, Newcastle-upon-Tyne (Vice-President, *Member of Council*)	Aug. 2, 1866

	Date of Election and of Transfer.
49 BERKLEY, C., Marley Hill, Swalwell, R.S.O., Co. Durham (Retiring VICE-PRESIDENT, *Member of Council*) ...	Aug. 21, 1852
50 BERKLEY, FREDERICK, Lumley Thicks, Fence Houses ...	A.M. Dec. 9, 1882
	M. June 8, 1889
51 BERKLEY, R. W., Marley Hill, Swalwell, R.S.O., Co. Durham	S. Feb. 14, 1874
	A.M. Aug. 7, 1880
	M. June 8, 1889
52 BEWICK, T. J., Broad Street House, Old Broad Street, London, E.C.	April 5, 1860
53 BEYNON, J. C. S., P.O. Box 1364, Johannesburg, Transvaal	June 10, 1893
54 BIGGE, D. SELBY, 27, Mosley Street, Newcastle-upon-Tyne...	June 13, 1891
55 BIGLAND, J., Henknowle, Bishop Auckland	June 3, 1857
56 BILGRAMI, SYED ALI SHAMSUL ULAMA, Director-General of Mines, Hyderabad, Deccan, India	April 11, 1891
57 BISHOP, JAMES, Grey Valley Coal Co., Brunnerton, Greymouth, New Zealand	April 13, 1889
58 BITZOS, N. J., c/o A. G. Sourlas, Balouk Bazaar, Constantinople	S. Feb. 14, 1891
	M. Apr. 14, 1894
59 BLACK, W., Hedworth Villa, South Shields	April 2, 1870
60 BLACKBURN, WM. STEVENSON, Aire Villas, Astley, Woodlesford, near Leeds	Dec. 10, 1887
61 BLACKETT, W. C., Jun., Acorn Close, Sacriston, Durham	S. Nov. 4, 1876
(Member of Council)	A.M. Aug. 1, 1885
	M. June 8, 1889
62 BLAKELEY, A. B., Soothill Wood Coll. Co., Ltd., near Batley	S. Feb. 15, 1879
	M. Aug. 3, 1889
63 BOLTON, EDGAR ORMEROD, Executor of Col. Hargreaves, Colliery Offices, Burnley	April 12, 1890
64 BOLTON, H. H., Newchurch Collieries, near Manchester ...	Dec. 5, 1868
65 BOUCHER, A. S., P.O. Box 53, Krugersdorf, South African Republic	Aug. 4, 1883
66 BOYD, WM., North House, Longbenton, Newcastle-upon-Tyne	Feb. 2, 1867
67 BRADFORD, GEO., Norton Palms, Darlington	Oct. 11, 1890
68 BRAMWELL, HUGH, Great Western Colliery, near Pontypridd, Glamorganshire	S. Oct. 4, 1879
	A.M. Aug. 6, 1887
	M. Aug. 3, 1889
69 BREAKELL, THOMAS, Brassington, near Derby	Feb. 11, 1893
70 BRECKON, J. R., 53, John Street, Sunderland	Sept. 3, 1864
71 BRITTEN, T. J., P.O. Box 494, Johannesburg, Transvaal	June 21, 1894
72 BROJA, RICHARD, Kœniglicher Oberbergrath, 35, Friédrich Strasse, Halle a/S. Germany	Nov. 6, 1880
73 BROUGH, BENNETT H., Grove Lodge, Thames Ditton, Surrey	A.M. Dec. 10, 1887
	M. June 8, 1889
74 BROUGH, THOMAS, New Seaham Colliery, Sunderland ...	S. Feb. 1, 1873
	A.M. Aug. 2, 1879
	M. June 8, 1889
75 BROWN, ARCHIBALD T., 372, Flinders Lane, Melbourne ...	Aug. 5, 1893
76 BROWN, M. WALTON, Westmorlands, Gateshead-upon-Tyne	S. Oct. 7, 1871
(SECRETARY)	M. Aug. 3, 1878

		Date of Election and of Transfer.
77	BROWN, ROBERT M., Norwood Colliery, *via* Darlington ...	A.M. Oct. 10, 1886
		M. Aug. 3, 1889
78	BROWN, THOMAS, District Manager, Westport Coal Co., Millerton, Westport, New Zealand	April 13, 1889
79	BROWN, THOS. FORSTER, Guildhall Chambers, Cardiff *(Member of Council)*	Aug. 1, 1861
80	BROWN, WESTGARTH FORSTER, Alston House, Parade, Cardiff	S. Aug. 6, 1887
		M. Aug. 5, 1893
81	BROWNE, SIR BENJAMIN C., Westacres, Benwell, Newcastle-upon-Tyne. Transactions to Free Library, Newcastle-upon-Tyne	Oct. 1, 1870
82	BROWNE, R. J., Barakar, East India Railway, Bengal ...	Oct. 8, 1892
83	BRUCE, JOHN, Port Mulgrave, Hinderwell, R.S.O., Yorkshire	S. Feb. 14, 1874
		A.M. Aug. 7, 1880
		M. June 8, 1889
84	BRYHAM, W., Douglas Bank Collieries, Wigan	Aug. 3, 1865
85	BULMAN, E. H., P.O. Box 357, Johannesburg, Transvaal ...	Feb. 13, 1892
86	BULMAN, H. F., Byer Moor, Burnopfield, near Newcastle-upon-Tyne	S. May 2, 1874
		A.M. Aug. 6, 1881
		M. June 8, 1889
87	BUNKELL, HENRY B., P.O. Box 962, Johannesburg, Transvaal	April 8, 1893
88	BUNNING, C. Z., c/o The Borax Co., Limited, 2, Macri Khan, Constantinople	S. Dec. 6, 1873
		A.M. Aug. 5, 1882
		M. Oct. 8, 1887
89*	BURLS, HERBERT T., Box 76, Barberton, Transvaal, South Africa	Feb. 9, 1889
90	BURN, JAMES, 28, Fawcett Street, Sunderland	Dec. 13, 1890
91*	BURNS, DAVID, Canal Bank, Carlisle	May 5, 1877
92	BURROWS, J. S., Green Hall, Atherton, near Manchester ...	S. Oct. 11, 1873
		M. Aug. 4, 1877
93	BUTTERS, CHARLES, P.O. Box 1891, Johannesburg, Transvaal	Feb. 10, 1894
94	CALVERT, A. F., The Mount, Oseney Crescent, Camden Road, London, N.W.	Apr. 14, 1894
95	CAMPBELL, W. YOUNG, P.O. Box 357, Johannesburg, Transvaal	Apr. 8, 1893
96*	CANDLER, T. E., East Lodge, Crook, *via* Darlington ...	S. May 1, 1875
		A.M. Aug. 4, 1883
		M. June 8, 1889
97	CARNES, CHARLES SPEARMAN, Hutton Henry Colliery, Wingate, R.S.O., Co. Durham	Aug. 1, 1891
98	CARR, T., Bengal Iron and Steel Coal Co., Limited, Colliery Department, Kendwah, Burrakur, East Indian Railway, Bengal, India	Feb. 10, 1894
99	CHALMERS, J. A., P.O. Box 357, Johannesburg, Transvaal	Dec. 9, 1893
100	CHAMBERS, A. M., Thorncliffe Iron Works, near Sheffield...	Mar. 6, 1869
101	CHAPMAN, ALFRED CRAWHALL, 29, St. Nicholas' Buildings, Newcastle-upon-Tyne...	Oct. 13, 1888
102	CHARLETON, A. G., Dashwood House, New Broad Street, London, E.C.	Aug. 6, 1892
103	CHARLTON, WM., Linares, Provincia de Jaen, Spain ...	April 8, 1893

		Date of Election and of Transfer.
104	CHATER, JOSEPH, Bengal Nagpur Coal Co., Limited, Evelyn Lodge, Kumarpur, Asansol, Bengal, India	Dec. 9, 1893
105	CHEESMAN, E. TAYLOR, Shire Moor Colliery, Northumberland	A. Aug. 2, 1890
		M. Aug. 6, 1892
106	CHEESMAN, HERBERT, Hartlepool	Aug. 6, 1892
107	CHEESMAN, I. T., Throckley Colliery, Newcastle-upon-Tyne	Feb. 1, 1873
108	CHEESMAN, W. T., Hartlepool	Feb. 5, 1876
109	CHICKEN, LANCELOT W., Boldon Colliery, Co. Durham ...	June 8, 1889
110	CHILDE, HENRY S., Mining Engineer, Wakefield	A.M. Feb. 12, 1887
		M. Aug. 3, 1889
111	CLARK, C. F., Garswood Coal and Iron Co., Ltd., near Wigan	Aug. 2, 1866
112	CLARK, R. B., Springwell Colliery, Gateshead-upon-Tyne ...	S. May 3, 1873
		M. Aug. 4, 1877
113	CLOUGH, JAMES, Willow Bridge, Choppington, Morpeth ...	S. April 5, 1873
		A.M. Aug. 3, 1878
		M. June 8, 1889
114	COCHRANE, B., Aldin Grange, Durham	Dec. 6, 1866
115	COCHRANE, C., Green Royde, Pedmore, near Stourbridge ...	June 3, 1857
116	COCHRANE, NAPIER, Aldin Grange, Durham	S. Dec. 8, 1888
		M. Feb. 10, 1894
117	COCHRANE, W., St. John's Chambers, Grainger Street West, Newcastle-upon-Tyne (PAST-PRESIDENT, *Member of Council*)	Aug. 1, 1861
118	COCKSON, CHARLES, Ince Coal and Cannel Co., Ince, Wigan	A.M. April 22, 1882
		M. Aug. 3, 1889
119	COGHLAN, F. M., Catorce, S.L.P., Mexico	Dec. 9, 1893
120	COLLINS, ARTHUR LAUNCELOT, 14 & 15, Broad Street Avenue, London, E.C.	Feb. 13, 1892
121*	COLLINS, H. B., Garscaddin, by New Kilpatrick, Dumbartonshire	April 14, 1894
122	COLQUHOUN, T., West Stanley Colliery, Co. Durham ...	Dec. 14, 1889
123	COOK, J., Washington Iron Works, Washington, Co. Durham	May 8, 1869
124	COOK, J. WATSON, Greenfield House, Crook	Oct. 14, 1893
125	CORBETT, V. W., Chilton Moor, Fence Houses	Sept. 3, 1870
126	CORBITT, M., Teams, Gateshead-upon-Tyne	Dec. 4, 1875
127	CORLETT, GEORGE STEPHEN, Wigan	Dec. 12, 1891
128	COULSON, F., 10, Victoria Terrace, Durham	S. Aug. 1, 1868
		M. Aug. 2, 1873
129	COULTHARD, SN. DN. FRANCISCO, Minas del Penoncillo, Marbella, Provincia de Malaga, Spain	June 8, 1889
130	COX, JOHN H., 10, St. George's Square, Sunderland	Feb. 6, 1875
131*	COXE, E. B., Drifton, Jeddo, P.O. Luzerne Co., Penna., U.S.	Feb. 1, 1873
132	CRAVEN, HIRAM, Jun., Mechanical Engineer, Sunderland ...	April 12, 1890
133	CRIGHTON, JOHN, 20, Exchange Buildings, St. Mary's Gate, Manchester	Oct. 8, 1887
134	CRONE, E. W., Forest Hall, near Newcastle-upon-Tyne ...	S. Mar. 5, 1870
		M. Aug. 5, 1876
135	CRONE, J. R., Tudhoe House, *via* Spennymoor	Feb. 1, 1868
136	CRONE, S. C., Forest Hall, Newcastle-upon-Tyne	Aug. 3, 1854
137	CROSS, JOHN, 77, King Street, Manchester	June 5, 1869
138	CROUDACE, C. J., Bettisfield Colliery Co., Limited, Bagillt, North Wales	Nov. 2, 1872

		Date of Election and of Transfer
139	CROUDACE, T., West House, Haltwhistle	Dec. 9, 1893
140	CROUDACE, THOMAS, Lambton Lodge, Lambton, Newcastle, New South Wales	Nov. 6, 1862
141	CURRY, W. THOS., Thornton House, Serpentine Road, Newport, Mon.	S. Sept. 4, 1880 A.M. Aug. 7, 1886 M. Aug. 3, 1889
142	DACRES, THOMAS, Six Bells Steam Coal Collieries, Aberbeeg, Monmouthshire	May 4, 1878
143	DAGLISH, JOHN, Rothley Lake, Cambo, R.S.O., Northumberland (PAST-PRESIDENT, *Member of Council*)	Aug. 21, 1852
144	DAKERS, W. R., The Loggins, Tudhoe Colliery, *via* Spennymoor, Durham	A.M. Oct. 14, 1882 M. Aug. 3, 1889
145	DALE, DAVID, West Lodge, Darlington	Feb. 5, 1870
146	DANGAR, J. HUBERT	April 8, 1893
147	DANIEL, PETER FRANCIS, Greymouth, New Zealand ...	April 8, 1893
148	DARLING, FENWICK, South Durham Colliery, Darlington ...	Nov. 6, 1875
149	DARLINGTON, JAMES, Black Park Colliery, Ruabon, North Wales	S. Nov. 7, 1874 M. Aug. 4, 1877
150	DAVEY, GEORGE, c/o Michoacan Railway and Mining Company, Las Trojes, Ocampo, Michoacan, Mexico ...	June 10, 1893
151	DAVEY, HY., 3, Princes Street, Westminster, London, S.W.	Oct. 11, 1873
152	DAVIES, Lt.-Col. JASPER G. S., Marton, Middlesbrough ...	Aug. 3, 1889
153	DAVIES, JOHN, Hartley House, Coundon, Bishop Auckland...	April 10, 1886
154	DAVIES J. HUBERT, P.O. Box 455, Johannesburg, Transvaal	April 8, 1893
155	DAVIS, KENNETH MCRAE, Dudley Colliery, Northumberland	April 13, 1889
156	DEES, J. GIBSON, Floraville, Whitehaven	Oct. 13, 1883
157	DEES, R. R., Newcastle-upon-Tyne	Oct. 7, 1871
158	DELPRAT, G. D., Campanas No. 15, Cordoba, Spain ...	Feb. 10, 1894
159	DENNISTON, ROBERT B., Stuart Street, Dunedin, New Zealand	A.M. Dec. 11, 1876 M. Aug. 3, 1889
160	DICKINSON, ARTHUR, Warham Road, South Croydon, Surrey	April 14, 1894
161	DIXON, D. W., Lumpsey Mines, Brotton, Saltburn-by-the-Sea	Nov. 2, 1872
162*	DIXON, JAMES S., 97, Bath Street, Glasgow	Aug. 3, 1878
163	DIXON, R., 4, Tynevale Terrace, Gateshead-upon-Tyne ...	June 5, 1875
164	DIXON-BROWN, A. D., c/o Messrs. Easton and Bessemer, Engineers, Taunton	Dec. 10, 1892
165	DOBBS, JOSEPH, Jarrow Colliery, Castle Comer, Co. Kilkenny	April 14, 1894
166	DODD, B., Bearpark Colliery, near Durham	S. May 3, 1866 M. Aug. 1, 1868
167	DODD, CYRIL H., St. Helens Colliery, Bishop Auckland ...	Aug. 6, 1892
168	DODD, M., Burncroft, Hexham	S. Dec. 4, 1875 A.M. Aug. 7, 1880 M. June 8, 1889
169*	DONKIN, W., c/o A. M. Appleten, 10, Western Hill, Durham	S. Sept. 2, 1876 A.M. Aug. 1, 1885 M. June 8, 1889
170	DOUGLAS, C. P., Parliament Street, Consett, Co. Durham ...	Mar. 6, 1869
171	DOUGLAS, JOHN, Seghill Colliery, Dudley, Northumberland	A.M. April 22, 1882 M. Aug. 3, 1889
172	DOUGLAS, M. H., Usworth Colliery, Washington, R.S.O., Co. Durham	A.M. Aug. 2, 1879 M. Aug. 3, 1889

		Date of Election and of Transfer.
173	DOUGLAS, T., The Garth, Darlington (PRESIDENT, *Member of Council)*	Aug. 21, 1852
174	DOWDESWELL, H., Butterknowle Colliery, *via* Darlington ...	S. April 5, 1873 M. Aug. 7, 1875
175	DOYLE, PATRICK, *Indian Engineering*, 19, Lall Bazar, Calcutta, India	A.M. Mar. 1, 1879 M. Aug. 3, 1889
176	DUDLEY, URIAH, P.O. Box 110, Broken Hill, New South Wales	April 14, 1894
177	DUMAT, ALFRED, 1, Loop Street, Pietermaritzburg, Natal	April 8, 1893
178	EASTLAKE, ARTHUR W., Balham, London	June 11, 1892
179	ECK, RICHARD, P.O. Box 1382, Johannesburg, Transvaal	Aug. 5, 1893
180	EDEN, C. H., c/o Messrs. Vivian and Sons, Swansea ...	June 14, 1890
181	EDGE, J. H., Coalport Wire Rope and Chain Works, Shifnal, Salop	A.M. Sept. 7, 1878 M. Aug. 3, 1889
182	ELLIS, W. R., Wigan	June 1, 1878
183	ELSDON, ROBERT, The Highlands, Burnt Ash Hill, Lee, Kent	Nov. 7, 1876
184	ELWEN, THOMAS LEE, Littleburn Colliery, near Durham ...	Oct. 13, 1888
185	EMBLETON, H. C., Central Bank Chambers, Leeds	April 14, 1894
186	EMBLETON, T. W., The Cedars, Methley, Leeds	S. Sept. 2, 1865 M. Aug. 1, 1868
187	EMINSON, J. B., Londonderry Offices, Seaham Harbour ...	Mar. 2, 1872
188	ETHERINGTON, J., 39A, King William Street, London Bridge, London, E.C.	Dec. 9, 1893
189	EVANS, E. W., P.O. Box 357, Johannesburg, Transvaal ...	April 8, 1893
190	EVANS, LEWIS, Coetzeestroom Estate and Gold Mining Co., Limited, Kaapsche Hoop, North Dekaap, Transvaal ...	Oct. 14, 1893
191	EVERARD, J. B., 6, Millstone Lane, Leicester	Mar. 6, 1869
192	FAIRLEY, JAMES, Craghead and Holmside Coll., Chester-le-Street	A.M. Aug. 7, 1880 M. Aug. 3, 1889
193	FARAGHER, LOUIS, Cape Copper Co., Limited, Ookiep, Namaqualand, South Africa	April 8, 1893
194	FARMER, A., Seaton Carew, near West Hartlepool	Mar. 2, 1872
195	FAULDER, JOSEPH, Bolton Colliery, Mealsgate, *via* Carlisle...	Aug. 6, 1892
196*	FENWICK, BARNABAS, 84, Osborne Road, Newcastle-upon-Tyne	Aug. 2, 1866
197	FERENS, FREDERICK J., Silksworth Colliery, Sunderland ...	S. Dec. 4, 1880 A.M. Aug. 4, 1888 M. Aug. 3, 1889
198	FERENS, F. S., 13, Railway Arches, Westgate Road, Newcastle-upon-Tyne	Oct. 14, 1893
199	FERGIE, CHAS., Drummond Colliery, Westville, Nova Scotia	Dec. 9, 1893
200	FERGUSON, D., 124, New City Road, Glasgow	A.M. Dec. 8, 1883 M. Aug. 3, 1889
201	FISHER, ED. R., Nant Glas, Cross Hands, near Llanelly, South Wales	A.M. Aug. 2, 1884 M. Aug. 3, 1889
202	FISHWICK, R., Binchester Colliery, Co. Durham	Aug. 6, 1892
203	FLEMING, C. E., Black, Hawthorn, and Co., Gateshead-upon-Tyne	April 13, 1889
204	FLETCHER, H., Ladyshore Colliery, Little Lever, Bolton, Lancashire	Aug. 3, 1865

		Date of Election and of Transfer.
205	FLETCHER, JAMES, Whickham and Bullock Island Coal Co., Limited, Carrington, New South Wales	Dec. 9, 1893
206	FLETCHER, LANCELOT, Brigham Hall, Carlisle	A.M. April 14, 1888
		M. June 8, 1889
207	FOGGIN, W., 16, Great George Street, Westminster, London, S.W.	Mar. 6, 1875
208	FORD, STANLEY H., c/o City and Suburban Gold Mining Co., P.O. Box 28, Johannesburg, Transvaal. Transactions, etc., to c/o F. G. Ford, 19, North Street, Bristol ...	June 10, 1893
209	FORREST, J. C., Holly Bank Coll., Essington, Wolverhampton	April 12, 1884
210	FORSTER, G. B., 3, Eldon Square, Newcastle-upon-Tyne (PAST-PRESIDENT, *Member of Council)*	Feb. 5, 1857
211	FORSTER, J. R., Water Company's Office, Newcastle-upon-Tyne	July 2, 1872
212	FORSTER, J. T.	S. Aug. 1, 1868
		M. Aug. 5, 1876
213	FORSTER, THOMAS E., 3, Eldon Square, Newcastle-upon-Tyne	S. Oct. 7, 1876
	(Member of Council)	A.M. Aug. 1, 1885
		M. June 8, 1889
214	FRYAR, MARK, Denby Colliery, Derby	S. Oct. 7, 1876
		A.M. Aug. 4, 1883
		M. June 8, 1889
215	GALLOWAY, T. LINDSAY, Argyll Colliery, Campbeltown, N.B.	Sept. 2, 1876
216	GALLOWAY, WM., Cardiff	April 23, 1887
217	GALLWEY, ARTHUR P., P.O. Box 138, Johannesburg, Z.A.R.	S. Oct. 2, 1880
		M. Oct. 10, 1891
218	GEDDES, GEORGE H., 142, Princes Street, Edinburgh ...	Oct. 1, 1881
219	GERRARD, JAMES, 11, Meek Buildings, Wigan	S. Mar. 1, 1873
		A.M. Aug. 2, 1879
		M. June 8, 1889
220	GERRARD, JOHN, Worsley, Manchester	S. Mar. 5, 1870
		M. Aug. 1, 1874
221	GIFFORD, HENRY J., 36, St. John's Road, Clifton, Bristol ...	Oct. 14, 1893
222	GILCHRIST, J. R., Garesfield Colliery, Lintz Green, Newcastle-upon-Tyne	S. Feb. 3, 1877
		A.M. Aug. 1, 1885
		M. June 8, 1889
223	GILLETT, LEONARD F., 163, Osmaston Road, Derby ...	June 14, 1890
224	GILROY, S. B., Markham Collieries, Duckmanton	Sept. 5, 1868
225	GJERS, JOHN, Southfield Villas, Middlesbrough	June 7, 1873
226	GLEDHILL, EDWARD, Carolina Hacienda, Honda, Republic of Colombia, South America	Dec. 9, 1893
227	GOOLDEN, WALTER T., 28, Westbourne Park, London, W. ...	Aug. 6, 1892
228	GOULDIE, JOSEPH, The North-Eastern Bullfontine, Kimberley, South Africa	Aug. 5, 1893
229	GREAVES, J. O., St. John's, Wakefield	Aug. 7, 1862
230	GREEN, J. T., Southside, Sneyd Park, Clifton, Bristol ...	Dec. 3, 1870
231	GREENER, HENRY, South Pontop Colliery, Annfield Plain...	A.M. Dec. 9, 1882
		M. Aug. 3, 1889
232	GREENER, T. Y., West Lodge, Crook, Darlington	S. July 2, 1872
		A.M. Aug. 2, 1879
		M. June 8, 1889

		Date of Election and of Transfer.
233	GREENWELL, G. C., Jun., Poynton, near Stockport	S. Mar. 6, 1869
		M. Aug. 3, 1872
234	GRESLEY, W. S., Erie, Pa., U.S.A.	A.M. Oct. 5, 1878
		M. Aug. 3, 1889
235	GREY, C. G., 20, Northbrook Road, Leeson Park, Dublin ...	May 4, 1872
236	GRIFFITH, N. R , Plasnewydd, Ruabon, North Wales ...	Aug. 1, 1867
237	GRIFFITH, WM., Waterloo House, Aberystwyth, South Wales	Dec. 9, 1893
238	GRIMSHAW, E. J., 23, Hardshaw Street, St. Helen's, Lancashire	Sept. 5, 1868
239	GUTHRIE, JAMES K., Eltringham Colliery, Prudhoe, R.S.O.	Aug. 3, 1889
240	HADDOCK, W. T., Jun., George and May Gold Mining Co., Limited, Krugersdorf, Transvaal	S. Oct. 7, 1876
		A.M. Aug. 1, 1885
		M. June 8, 1889
241	HAGGIE, D. H., Wearmouth Patent Rope Works, Sunderland	Mar. 4, 1876
242	HAGGIE, PETER SINCLAIR, Gateshead-upon-Tyne	A.M. April 14, 1883
		M. Aug. 3, 1889
243*	HAGUE, ERNEST, Castle Dyke, Sheffield	Aug. 3, 1878
244	HALDER, ALBERT H., Pietersburg, Transvaal	April 8, 1893
245	HALL, FREDERICK, Fernleigh, Highfield, Workington ...	Oct. 14, 1893
246	HALL, FRED. W., Haswell Lodge, Sunderland	S. June 8, 1889
		M. Feb. 10, 1894
247	HALL, M., Lofthouse Station Collieries. near Wakefield ...	Sept. 5, 1868
248	HALL, M. S., 8, Victoria Street, Bishop Auckland	Feb. 14, 1874
249	HALL, TOM, Ryhope Colliery, *via* Sunderland	June 8, 1889
250	HALL, WILLIAM F., Haswell Colliery, Haswell, *via* Sunderland (VICE-PRESIDENT, *Member of Council)*	May 13, 1858
251	HALLAS, G. H., Wigan and Whiston Coal Co., Limited, Prescot	S. Oct. 7, 1876
		A.M. Aug. 4, 1883
		M. June 8, 1889
252	HALLIMOND, WM. TASKER, P.O. Box 344, Johannesburg, Transvaal	Dec. 14, 1889
253	HALSE, EDWARD, Apartado, 512, Mexico, D.F.	A.M. June 13, 1885
		M. Aug. 3, 1889
254	HAMILTON, E., Rig Wood, Saltburn-by-the-Sea	S. Nov. 1, 1873
		A.M. Aug. 2, 1879
		M. June 8, 1889
255	HANN, EDMUND, Aberaman, Aberdare	S. Sept. 5, 1868
		M. Aug. 3, 1872
256	HARDMAN, JOHN E., 83, Hollis Street, Halifax, Nova Scotia	Dec. 9, 1893
257	HARE, SAMUEL, Bedlington Collieries, Bedlington, R.S.O....	S. Aug. 2, 1879
		M. Aug. 1, 1891
258	HARGREAVES, JOSEPH, Gwaun Cae Gurwen Collieries, Brynamman, R.S.O.	Aug. 5, 1893
259	HARGREAVES, WILLIAM, Rothwell Haigh, Leeds	Sept. 5, 1868
260	HARLE, PETER, Page Bank Colliery, Co. Durham	Oct. 8, 1892
261	HARLE, RICHARD, Browney Colliery, Durham	April 7, 1877
262	HARRIS, G. E., Margherita, Debrugarh, Upper Assam ...	Feb. 10, 1894
263	HARRIS, W. S., Kibblesworth, Gateshead-upon-Tyne ...	S. Feb. 14, 1874
		A.M. Aug. 7, 1880
		M. June 8, 1889

		Date of Election and of Transfer.
264	HARRISON, G. B., Swinton, near Manchester	Aug. 6, 1892
265	HARRISON, W. B., Brownhills Collieries, near Walsall ...	April 6, 1867
266	HAY, J., Jun., Widdrington Colliery, Acklington	S. Sept. 4, 1869
		M. Aug. 4, 1874
267	HEAD, JEREMIAH, 47, Victoria Street, Westminster, London, S.W.	Aug. 3, 1889
268	HEDLEY, J. L., H.M. Inspector of Mines, 22, Hawthorn Terrace, Newcastle-upon-Tyne (VICE-PRESIDENT, *Member of Council*)	S. Feb. 5, 1870
		M. Aug. 2, 1873
269	HEDLEY, SEPT. H., Bank Chambers, Wakefield	S. Feb. 15, 1879
		A.M. Aug. 1, 1885
		M. Aug. 3, 1889
270	HEDLEY, W. H., Medomsley, R.S.O., Co. Durham (*Member of Council*)	Aug. 3, 1865
271	HENDERSON, H., Pelton Colliery, Chester-le-Street	Feb. 14, 1874
272	HENDERSON, JOS. J., U.S. Engineer's Office, Kingsbridge, New York, U.S.A.	June 13, 1891
273	HENDY, J. C. B., Etherley, *via* Darlington	Oct. 14, 1893
274	HENSHAW, A. MAYON, Talk-o'-th'-Hill Coll, Stoke-upon-Trent	June 8, 1889
275	HEPPELL, T., Leafield House, Birtley, Chester-le-Street (*Member of Council*)...	Aug. 6, 1863
276	HESLOP, C., Upleatham and Lingdale Mines, Upleatham, R.S.O., Yorkshire	S. Feb. 1, 1868
		M. Aug. 2, 1873
277	HESLOP, GRAINGER, Deptford Hall, Sunderland	Oct. 5, 1872
278	HESLOP, MICHAEL, Rough Lea Colliery, Willington, Co. Durham	A. Feb. 10, 1894
		M. June 21, 1894
279	HESLOP, THOMAS, Storey Lodge Colliery, Cockfield, *via* Darlington	S. Oct. 2, 1880
		A.M. Aug. 4, 1888
		M. Aug. 3, 1889
280*	HEWITT, G. C., Coal Pit Heath Colliery, near Bristol ...	June 3, 1871
281	HEWLETT, A., Haseley Manor, Warwick	Mar. 7, 1861
282	HIGBY, ROBERT GEORGE, Borrea Coal Co., Limited, Sitarampore, Bengal, East India Railway	Dec. 9, 1893
283	HIGSON, JACOB, 18, Booth Street, Manchester	Aug. 7, 1862
284	HILL, WILLIAM, The Pooley Hall Colliery Co., Polesworth, near Tamworth	A.M. June 9, 1883
		M. Aug. 3, 1889
285*	HILTON, J., 67, Hawkshead Street, Southport	S. Dec. 7, 1867
		M. Aug. 6, 1870
286	HILTON, T. W., Wigan Coal and Iron Co., Limited, Wigan	Aug. 3, 1865
287	HODGSON, JOHN, Edmondsley Colliery, Chester-le-Street ...	Oct. 8, 1889
288	HOLLIDAY, MARTIN F., Langley Grove, Durham	May 1, 1875
289	HOLLIS, HENRY WM., Whitworth House, Spennymoor ...	June 8, 1889
290	HOLMES, C., Grange Hill, near Bishop Auckland	April 11, 1874
291	HOLT, JOHN, Jun., 109, Manchester Road, Swinton, Lancashire, and The Hollies, Heywood, Lancashire	Aug. 5, 1893
292	HOOD, A., 6, Bute Crescent, Cardiff	May 2, 1861
293	HOOPER, ED., c/o J. H. Hooper, College Precincts, Worcester	A.M. June 4, 1881
		M. April 14, 1894
294	HOPKINS, EDWARD, 13, Harrington Gardens, London, S.W.	Oct. 8, 1889
295	HOSKOLD, H. D., Inspector General of Mines of the Argentine Republic, and Director of the National Department of Mines and Geology, Casilla Correos, 900, Buenos Ayres	April 1, 1871

	Date of Election and of Transfer.
296 Houlgate, J. Kerr, 69, Lowther Street, Whitehaven ...	April 8, 1893
297 Howes, Frank T., Singareni Collieries, Hyderabad Deccan Co., Secunderabad, India	A. Dec. 10, 1892 M. Oct. 14, 1893
298 Humble, William, Coal-fields Office, Newcastle, New South Wales	Oct. 14, 1893
299 Hurst, George, 58, Eldon Street, Newcastle-upon-Tyne ...	S. April 14, 1883 M. Aug. 1, 1891
300 Jackson, W. G., Hicklam House, Aberford, near Leeds ...	June 7, 1873
301 Jeffcock, T. W., 18, Bank Street, Sheffield	Sept. 4, 1869
302 Jenkins, W., Ocean Collieries, Treorky, Glamorganshire ...	Dec. 6, 1862
303 Jenkins, Wm , Consett Iron Works, Consett, Durham ...	May 2, 1874
304 Jepson, H., 20, The Avenue, Durham	S. July 2, 1872 A.M. Aug. 2, 1879 M. June 8, 1889
305*Jobling, Thos. E., Croft Villa, Blyth, Northumberland ...	S. Oct. 7, 1876 A.M. Aug. 4, 1883 M. June 8, 1889
306 Johns, J. Harry, P.O. Box 1021, Johannesburg, Transvaal	June 21, 1894
307 Johnson, J., The Terrace, Stairfoot, near Barnsley ...	Mar. 7, 1874
308 Johnson, W., Abram Colliery, Wigan	S. Feb. 14, 1874 A.M. Aug. 2, 1879 M. June 8, 1889
309 Johnson, Wm., Radcliffe Colliery, Acklington, Northumberland	Dec. 9, 1882
310 Johnston, J. Howard, c/o Messrs. Backus and Johnston, Lima, Peru, South America	Feb. 10, 1894
311 Joicey, J. G., Forth Banks West Factory, Newcastle-upon-Tyne	April 10, 1869
312 Joicey, W. J., Sunningdale Park, Berkshire	Mar. 6, 1869
313 Jones, Jacob Carlos, Bellambi, New South Wales	Aug. 6, 1892
314 Jones, John Arthur, Gijon, Asturias, Spain...	April 8, 1893
315 Kayll, A. C., Gosforth, Newcastle-upon-Tyne	S. Oct. 7, 1876 M. Aug. 3, 1889
316 Kellett, William, Portland Bank, Southport	June 1, 1878
317 Kendall, John D., Foxhouses Road, Whitehaven ...	Oct. 3, 1874
318 Kirkby, J. W., Kirkland, Leven, Fife	Feb. 1, 1873
319 Kirkup, J. P., Burnhope, Lanchester	April 11, 1891
320 Kirkup, Philip, Cornsay Colliery Office, Esh, near Durham	S. Mar. 2, 1878 A.M. Aug. 7, 1886 M. Aug. 3, 1889
321 Kirton, Hugh, Kimblesworth Colliery, Chester-le-Street ...	S. April 7, 1877 A.M. Aug. 1, 1885 M. June 8, 1889
322*Knowles, Robert, Ednaston Lodge, near Derby	April 10, 1886
323 Lalor, John O'Gorman, P.O. Box 357, Johannesburg, Transvaal	April 8, 1893
324 Lamb, R., Troughton House, Cleator Moor, *via* Carnforth ..	Sept. 2, 1865
325 Lamb, R. O., West Denton, Newcastle-upon-Tyne	Aug. 2, 1866
326 Lamb, Richd. W., 29, Great Cumberland Place, London, W.	Nov. 2, 1872

		Date of Election and of Transfer.
327	LANCASTER, JOHN, Ashlawn, Rugby	Mar. 2, 1865
328	LANCASTER, JOHN, Auchinheath, Southfield and Fence Collieries, Lesmahagow	Sept. 7, 1878
329	LANDALE, A., Comely Park Place, Dunfermline	Dec. 2, 1858
330*	LAPORTE, H., 57, Rue de la Concorde, Brussels	May 5, 1877
331	LAVERICK, JOHN WALES, Tow Law Colliery Office, Tow Law, R.S.O., Co. Durham	A.M. Dec. 9, 1882 M. Aug. 3, 1889
332	LAVERICK, ROBT., West Rainton, Fence Houses	Sept. 2, 1876
333	LAWFORD, CHAS. E., P.O. Box 238, Johannesburg, Transvaal	June 21, 1894
334	LAWRENCE, H., Grange Iron Works, Durham *(Member of Council)*	Aug. 1, 1868
335	LAWRENCE, H. L., c/o Dr. Magin, P.O. Box 733, Johannesburg, Transvaal	April 8, 1893
336	LAWS, W. G., Town Hall, Newcastle-upon-Tyne	Oct. 2, 1880
337	LEACH, C. C., Seghill Colliery, Northumberland *(Member of Council)*	S. Mar. 7, 1874 A.M. Aug. 6, 1881 M. Aug. 4, 1883
338	LEBOUR, G.A., Durham College of Science, Newcastle-upon-Tyne	Feb. 1, 1873
339	LECKIE, ROBERT G., Londonderry Iron Co., Limited, Londonderry, Nova Scotia	Dec. 9, 1893
340	LEE, JOHN F., Sheepbridge, Chesterfield	A.M. June 13, 1885 M. Aug. 3, 1889
341	LEE, J. W. R., Potes, Provincia de Santander, Spain ...	Aug. 5, 1893
342	LEWIS, H. R., 7, Drapers Gardens, London, E.C.	April 14, 1894
343	LEWIS, SIR WILLIAM THOMAS, Mardy, Aberdare *(Member of Council)*	Sept. 3, 1864
344	LIDDELL, J. M., 3, Victoria Villas, Newcastle-upon-Tyne ...	S. Mar. 6, 1875 A.M. Aug. 6, 1881 M. June 8, 1889
345	LIEBERT, R. A. D., P.O. Box 357, Johannesburg, Transvaal	April 8, 1893
346	LINDAY, GEORGE, Blackett Colliery, Haltwhistle	June 10, 1893
347	LINDAY, JAMES, Bishop Auckland	Aug. 6, 1892
348	LINDOP, A. B., Blackball, *via* Greymouth, New Zealand ...	Dec. 9, 1893
349	LINSLEY, R., Cramlington Colliery, Northumberland ...	July 2, 1872
350	LINSLEY, S. W., Whitburn Colliery, South Shields	Sept. 4, 1869
351	LISHMAN, R. R., Durham Main Colliery, Durham	S. June 9, 1883 M. Aug. 1, 1891
352	LISHMAN, T., Hetton Colliery, Hetton-le-Hole, R.S.O. ...	S. Nov. 5, 1870 M. Aug. 3, 1872
353	LISHMAN, WM., Holly House, Witton-le-Wear	April 1, 1858
354	LISHMAN, WM., Bunker Hill, Fence Houses	Mar. 7, 1861
355	LISHMAN, W. ERNEST, Bunker Hill, Fence Houses	June 10, 1893
356	LISLE, J., The Ottos Kopje Diamond Mines, Limited, Ottos Kopje Chambers, P.O. Box 381, Kimberley, South Africa	S. July 2, 1872 A.M. Aug. 3, 1878 M. June 8, 1889
357	LIVEING, E. H., 52, Queen Anne Street, Cavendish Square, London, W.	S. Sept. 1, 1877 A.M. Aug. 2, 1884 M. Aug. 3, 1889
358	LIVESEY, C., Bradford Colliery, near Manchester	Aug. 3, 1865
359	LIVESEY, T., Bradford Colliery, near Manchester	S. Nov. 7, 1874 M. Aug. 7, 1875

			Date of Election and of Transfer.
360	Llewellin, David Morgan, Glanwern Offices, Pontypool		May 14, 1881
361	Lloyd, G. Hamilton, Dolgerddon, Rhayader, Radnorshire. Transactions, etc., to 9, Lower Belgrave Street, London, S.W.		Dec. 9, 1893
362	Logan, William, Langley Park Colliery, Durham		Oct. 5, 1867
363	Longbotham, Jonathan, Barrow Collieries, Barnsley, Yorkshire	S.	May 2, 1868
		M.	Aug. 2, 1873
364	Longbotham, R. H., Westgate, Wakefield		April 8, 1893
365	Longridge, Jethro, Coxlodge Colliery, Newcastle-upon-Tyne		Dec. 14, 1889
366	Louis, D. A., 77, Shirland Gardens, London, W.		April 8, 1893
367	Lowdon, Thos., Hamsteels, near Durham		Dec. 14, 1889
368	Lupton, A., 6, De Grey Road, Leeds		Nov. 6, 1869
369	MacArthur, John S., 12, Knowe Terrace, Pollokshields, Glasgow		April 8, 1893
370	MacCabe, H. O., Russell Vale, Wollongong, New South Wales	S.	Sept. 7, 1878
		A.M.	Aug. 7, 1880
		M.	Aug. 3, 1889
371	McCarthy, E. T., c/o Col. Pigott, Archer Lodge, Charles Road, St. Leonards-on-the-Sea	A.M.	Oct. 8, 1887
		M.	Aug. 3, 1889
372	McCreath, J., 203, St. Vincent Street, Glasgow		Mar. 5, 1870
373	McCulloch, David, Beech Grove, Kilmarnock, N.B. ...		Dec. 4, 1875
374	McLaren, B., Jun., The Tower, Bedlington	S.	Dec. 8, 1883
		A.M.	Aug. 4, 1888
		M.	Aug. 3, 1889
375	McMurtrie, George Edwin James, Foxes Bridge Colliery, Cinderford, Gloucestershire	S.	Aug. 2, 1884
		M.	Dec. 12, 1891
376	McMurtrie, J., Radstock Colliery, Bath		Nov. 7, 1863
377	Maddison, Thos. R., Dirtcar House, near Wakefield ...	S.	Mar. 3, 1877
		A.M.	Aug. 6, 1881
		M.	June 8, 1889
378	Maddison, W. H. F., The Lindens, Darlington		June 14, 1890
379	Maling, C. T., Ellison Place, Newcastle-upon-Tyne ...		Oct. 5, 1872
380	Mammatt, J. E., St. Andrew's Chambers, Leeds		Aug. 3, 1865
381	Markham, G. E., Coundon, Bishop Auckland	S.	Dec. 4, 1875
		A.M.	Aug. 7, 1880
		M.	June 8, 1889
382	Marley, J. W., Thornfield, Darlington	S.	Aug. 1, 1868
		M.	Aug. 2, 1873
383	Marten, E. B., Pedmore, near Stourbridge		July 2, 1872
384	Martin, C. W., Murton Colliery, *via* Sunderland		Aug. 6, 1892
385	Mathieson, Alexander, Hetton Colliery, Carrington, near Newcastle, New South Wales		Nov. 5, 1892
386	Matthews, J., Messrs. R. and W. Hawthorn, Newcastle-upon-Tyne	A.M.	April 11, 1885
		M.	Aug. 3, 1889
387	Matthews, R. F., Harehope Hall, Alnwick		Mar. 5, 1857
388	Maughan, J. A., Government Central Provinces' Collieries, Umaria, *via* Katni, C.P., India	S.	Nov. 7, 1863
		M.	Aug. 6, 1870
389	Mawson, R. Bryham, Brick House, Westleigh, near Manchester		June 11, 1892

	Date of Election and of Transfer.
390 May, G., Harton Colliery Offices, near South Shields (Vice-President, *Member of Council*)	Mar. 6, 1862
391 Merivale, Prof. J. H., 2, Victoria Villas, Newcastle-upon-Tyne *(Member of Council)*...	May 5, 1877
392*Merivale, W., General Manager, Barbados Railway, Barbados	A.M. Mar. 5, 1881 M. Aug. 3, 1889
393 Merritt, W. Hamilton, 485, Huron Street, Toronto, Ontario, Canada	Oct. 14, 1893
394 Metcalf, A. T., Darkton, *via* (Ermelo and Lake Chrissie, Transvaal) Swaziland	June 21, 1894
395 Meysey-Thompson, A. H., Sun Foundry, Leeds	A.M. April 3, 1889 M. Aug. 3, 1889
396 Middleton, Robert, Sheep Scar Foundry, Leeds	Aug. 1, 1891
397*Miller, N.	A.M. Oct. 5, 1878 M. Aug. 3, 1889
398 Mitchell, Chas., Jesmond, Newcastle-upon-Tyne	April 11, 1874
399 Mitchell, Clayton E. J., West Highlands, Winchester ...	Nov. 5, 1892
400 Mitchell, Joseph, Mining Offices, Eldon Street, Barnsley	Feb. 14, 1874
401 Mitchinson, R., Pontop Colliery, Annfield Plain, R.S.O., Co. Durham	Feb. 4, 1865
402 Mitton, A. Dury, Marlpool House, near Derby	Aug. 6, 1892
403 Monkhouse, Jos., Gilcrux, Carlisle	June 4, 1863
404 Moor, Wm., Ocean House, Hartlepool	S. July 2, 1872 M. Aug. 4, 1877
405 Moore, R. T., 156, St. Vincent Street, Glasgow	Oct. 8, 1892
406 Moore, R. W., Somerset House, Whitehaven	S. Nov. 5, 1870 M. Aug. 4, 1877
407 Moore, William, Loftus Mines, Loftus-in-Cleveland, R.S.O.	A.M. Nov. 19, 1881 M. Aug. 3, 1889
408 Moreing, C. A., Broad Street House, Old Broad Street, London, E.C.	S. Nov. 7, 1874 A.M. Aug. 6, 1881 M. Aug. 3, 1889
409 Morison, John, Newbattle Collieries, Dalkeith, N.B. ...	A.M. Dec. 4, 1880 M. Aug. 3, 1889
410 Morris, Wm., Waldridge Colliery, Chester-le-Street ...	Oct. 8, 1892
411*Morton, H. J., 2, Westbourne Villas, South Cliff, Scarborough	Dec. 5, 1856
412 Mountain, William C., Forth Banks, Newcastle-upon-Tyne	April 9, 1892
413 Mulholland, M. L., West Cornforth, R.S.O., Co. Durham	A.M. Dec. 11, 1886 M. Aug. 3, 1889
414 Mundle, Arthur, St. Nicholas' Chambers, Newcastle-upon-Tyne	S. June 5, 1875 M. Aug. 4, 1877
415 Murton, Chas. J., Delaval Benwell Colliery, Newcastle-upon-Tyne	S. Mar. 6, 1880 A.M. Aug. 3, 1887 M. Aug. 3, 1889
416*Nasse, Rudolph, Geheimerbergrat, Dörnsbergstrasse, 6, Berlin, W., Germany	Sept. 4, 1869
417 Nevin, John, Littlemoor, Mirfield, Normanton	S. May 2, 1868 M. Aug. 5, 1871
418 Newton, James, Whitehaven Colliery, Whitehaven ...	Nov. 5, 1892
419 Nichol, Wm., De Beers Mine, Kimberley, South Africa ...	A.M. Oct. 9, 1886 M. Aug. 3, 1889

		Date of Election and of Transfer.
420	NICHOLLS, Captain G., H.M. Commissioner of Mines for Natal and Zululand, Pietermaritzburg, Natal	Oct. 14, 1893
421	NICHOLSON, J. COOK, 59, Side, Newcastle-upon-Tyne ...	Feb. 10, 1894
422	NICHOLSON, J. H., Cowpen Colliery Office, Blyth	S. Oct. 1, 1881
		A. Aug. 3, 1889
		M. April 8, 1893
423	NICHOLSON, MARSHALL, Middleton Hall, Leeds	Nov. 7, 1863
424	NIXON, RALPH, 47, Holly Avenue, Newcastle-upon-Tyne ...	Oct. 14, 1893
425	NORTH, F. W., Princes Chambers, Corporation Street, Birmingham	Oct. 6, 1864
426	OATES, ROBERT J. W., Newholme, Park Road, St. Anne's-on-the-Sea, Lancashire	S. Feb. 10, 1883
		A.M. Aug. 1, 1891
		M. Dec. 12, 1891
427	OGILVIE, A. GRAEME, 8, Grove End Road, St. John's Wood, London	Mar. 3, 1877
428	OLDHAM, GEORGE, 25, Western Hill, Durham...	Nov. 5, 1892
429	ORNSBY, R. E., Seaton Delaval Colliery, Newcastle-upon-Tyne	S. Mar. 6, 1875
		A.M. Aug. 4, 1883
		M. June 8, 1889
430	PALMER, SIR CHAS. MARK, Bart., M.P., Quay, Newcastle-upon-Tyne	Nov. 5, 1852
431	PALMER, A. S., Field House, Gateshead-on-Tyne	July 2, 1872
432	PALMER, HENRY, East Howle Colliery, near Ferryhill *(Member of Council)*	S. Nov. 2, 1878
		A.M. Aug. 4, 1883
		M. Aug. 3, 1889
433	PAMELY, C., 21, Morgan Street, Pontypridd, South Wales	S. Sept. 5, 1868
		M. Aug. 5, 1877
434	PANTON, F. S., Silksworth Colliery, Sunderland	S. Oct. 5, 1867
		M. Aug. 4, 1874
435	PARRINGTON, M. W., Wearmouth Colliery, Sunderland ...	S. Dec. 1, 1864
		M. Aug. 6, 1870
436	PARSONS, HON. CHARLES ALGERNON, Elvaston Hall, Ryton-upon-Tyne	A.M. June 12, 1886
		M. Aug. 3, 1889
437	PEAKE, R. C., Cumberland House, Redbourn, Herts. ...	S. Feb. 7, 1880
		A.M. Aug. 7, 1886
		M. Aug. 3, 1889
438	PEASE, SIR J. W., Bart., M.P., Hutton Hall, Guisbrough, Yorkshire	Mar. 5, 1857
439	PEEL, ROBERT, New Brancepeth Colliery, Durham	Aug. 6, 1892
440	PEILE, WILLIAM, Croft Hill, Moresby, Whitehaven ...	S. Oct. 1, 1863
		M. Aug. 6, 1870
441	PICKUP, P. W., 71, Preston New Road, Blackburn ...	Feb. 6, 1875
442	PLUMMER, JOHN, H.M. Inspector of Mines, Bishop Auckland	June 8, 1889
443	POTTER, A. M., Aydon Grange, Corbridge-upon-Tyne *(Member of Council)*	S. Feb. 3, 1872
		M. Aug. 5, 1876
444	POTTER, C. J., Heaton Hall, Newcastle-upon-Tyne	Oct. 3, 1874
445	POTTS, JOS., Jun., North Cliff, Roker, Sunderland	Dec. 6, 1879
446	PRICE, S. R., c/o Messrs. Forster Brown and Rees, 12, Victoria Street, Westminster, London, S.W.	S. Nov. 3, 1877
		A.M. Aug. 1, 1885
		M. Aug. 3, 1889

		Date of Election and of Transfer.
447	Pringle, Edward, Choppington Colliery, Northumberland	Aug. 4, 1877
448	Pringle, Thomas, Tanfield Lea Colliery, Lintz Green ...	Aug. 6, 1892
449*	Prior, Edward G., Victoria, British Columbia	Feb. 7, 1880
450	Pritchett, C. W., Jun., Apartado 84, Pachuca, Estado Hidalgo, Mexico	Dec. 9, 1893
451	Proctor, C. P.	S. Oct. 7, 1876
		A.M. Aug. 1, 1885
		M. June 8, 1889
452	Prout, James, 16, Claremont Road, Redruth, Cornwall ...	Feb. 10, 1894
453	Proust, Rene, Mammoth Hydraulic Sluicing Company, Matakitaki River, Central Buller, New Zealand ...	June 21, 1894
454	Quirk, J. S., St. Helens Lead Smelting Works, St. Helens, Lancashire	June 10, 1893
455	Ramsay, J. A., Sherburn and Littletown Collieries, near Durham	Mar. 6, 1869
456	Ramsay, Wm., Tursdale, Co. Durham	Sept. 11, 1875
457	Redmayne, R. A. S., Seaton Delaval Colliery, Newcastle-upon-Tyne	S. Dec. 13, 1884
		M. Dec. 12, 1891
458	Reid, Andrew, Newcastle-upon-Tyne	April 2, 1870
459	Reid, A. H., P.O. Box 746, Johannesburg, Transvaal ...	June 21, 1894
460	Reid, Francis, 13, Railway Arches, Westgate Road, Newcastle-upon-Tyne	April 9, 1892
461	Reid, P. S., 20, John Street, Adelphi, London, W.C. ...	Dec. 10, 1892
462	Renwick, T. C., Manor House, Lumley Park, Fence Houses	April 14, 1894
463	Rhodes, C. E., Car House, Rotherham	Aug. 4, 1883
464	Rhodes, F. B. F., Superintendent of the National Smelting and Refining Co., South Chicago, Illinois, U.S.A. ...	Feb. 10, 1894
465	Rich, William, Minas de Rio Tinto, Provincia de Huelva, Spain	A.M. June 9, 1888
		M. Aug. 3, 1889
466	Richardson, H., Backworth Colliery, Newcastle-upon-Tyne *(Member of Council)*	Mar. 2, 1865
467	Richardson, Ralph, Whitburn Colliery, South Shields ...	S. June 9, 1883
		M. Aug. 1, 1891
468	Ridley, G., 16, Dean Street, Newcastle-upon-Tyne... ...	S. Feb. 4, 1865
		M. Aug. 1, 1868
469	Ridyard, J., Bridgewater Offices, Walkden, near Bolton-le-Moors, Lancashire	Nov. 7, 1874
470	Ritson, U. A., 15, Queen Street, Newcastle-upon Tyne ...	Oct. 7, 1871
471	Robertson, D. A. W., Metropolitan Colliery, Helensburgh, near Sydney, New South Wales	Aug. 6, 1892
472*	Robertson, J. R. M., Linton, Mitson's Point, Sydney, New South Wales	Aug. 2, 1890
473	Robertson, W., 123, St. Vincent Street, Glasgow	Mar. 5, 1870
474	Robinson, C., P.O. Box 1244, Johannesburg, Transvaal ...	Dec. 9, 1893
475	Robinson, G. C., Brereton and Hayes Colliery, Rugeley, Staffordshire	Nov. 5, 1870
476	Robinson, J. B., Hedley Hill, Colliery, Waterhouses ...	Aug. 5, 1893
477	Robinson, John Thomas, South Medomsley Colliery, Dipton, R.S.O.	Feb. 13, 1892

		Date of Election and of Transfer.
478	Robinson, R., Howlish Hall, near Bishop Auckland *(Member of Council)*	Feb. 1, 1868
479	Robson, J. S., Butterknowle Colliery, *via* Darlington ...	May 15, 1862
480	Robson, T. O., Chowdene Cottage, Low Fell *(Member of Council)*	S. Sept. 11, 1875 A.M. Aug. 2, 1884 M. June 8, 1889
481	Rochester, William, River View, Blaydon-upon-Tyne ...	Dec. 13, 1891
482	Ronaldson, J. H., Mount Kembla Colliery, Wollongong, New South Wales	Aug. 6, 1892
483	Roscamp, J., Shilbottle Colliery, Lesbury, R.S.O., Northumberland	Feb. 2, 1867
484	Ross, Hugh, Croxdale Colliery Office, Durham	Aug. 6, 1892
485	Ross, J. A. G., Newcastle-upon-Tyne	July 2, 1872
486	Rosser, W., Rhydyrhelig, Sketty, Swansea	Aug. 6, 1857
487	Rothwell, R. P., 27, Park Place, New York, U.S.A. ...	Mar. 5, 1870
488	Routledge, Jos., Greenbank, Chester-le-Street	Sept. 11, 1875
489	Routledge, W. H., The Rhyd, Tredegar, Mon., Wales ...	S. Oct. 7, 1876 A.M. Aug. 1, 1885 M. June 8, 1889
490	Rowan, James, Inspector of Collieries, Wollongong, New South Wales	Aug. 6, 1892
491	Rowley, Walter, 20, Park Row, Leeds	Aug. 5, 1893
492	Russell, Robert, Coltness Iron Works, Newmains, N.B....	Aug. 3, 1878
493	Rutherford, W., South Derwent Colliery, Annfield Plain, Lintz Green	Oct. 3, 1874
494	Ryder, W. J. H., Messrs. Mills and Sons, Collingwood Street, Newcastle-upon-Tyne	Nov. 4, 1876
495*	Saise, Walter, Manager, E.I.R. Collieries, Giridih, Bengal, India	A.M. Nov. 3, 1877 M. Aug. 3, 1889
496	Sam, T. B. F., c/o Messrs. F. and H. Swanzy, 147, Cannon Street, London, E.C.	Aug. 5, 1893
497	Samborne, John Stukely Palmer, The Llandyry Anthracite Colliery Co., Limited, near Kidwelly, South Wales	Aug. 1, 1891
498	Sawyer, A. R., P.O. Box 1331, Johannesburg, Transvaal ...	S. Dec. 6, 1873 A.M. Aug. 2, 1879 M. June 8, 1889
499	Scarth, W. T., Raby Castle, Staindrop, Darlington ...	April 4, 1868
500	Scott, C. F., Grove Cottage, Leadgate, Co. Durham ...	S. April 11, 1874 M. Aug. 4, 1877
501	Scott, Ernest, Close Works, Newcastle-upon-Tyne ...	April 9, 1892
502	Scott, Joseph, Newcastle Street, Stockton, near Newcastle, New South Wales	Aug. 5, 1893
503	Scott, Joseph S., Trimdon Hall, Trimdon Grange, R.S.O....	S. Nov. 19, 1881 M. April 9, 1892
504	Scoular, G., Cleator Moor, *via* Carnforth	July 2, 1872
505	Seccombe, Alfred F., Albaston, Gunnislake, Cornwall ...	Dec. 9, 1893
506	Selby, Atherton, Leigh, near Manchester	Oct. 13, 1883
507	Selkirk, John Gunsen, Dalton-in-Furness	April 8, 1893
508	Seymour, L. Irving, De Beers Consolidated Mines, Limited, Kimberley, South Africa	Feb. 14, 1891

		Date of Election and of Transfer.
509	Shaw, F. George, c/o Bainbridge, Seymour, and Co., 13, St. Helen's Place, London, E.C.	June 10, 1893
510	Shaw, J. Leslie, Somerset House, Whitehaven	Oct. 8, 1892
511	Shiel, John, Framwellgate Colliery, Co. Durham	May 6, 1871
512	Shipley, T., 26, Riversdale Terrace, Sunderland	A.M. Aug. 2, 1884
		M. Aug. 3, 1889
513	Shore, Wm. Martin, Kaitangata Railway and Coal Co.'s Collieries, Otago, New Zealand	A.M April 13, 1889
		M. Aug. 3, 1889
514	Shotton, John, Ottoman Railway Co., Smyrna, Asia Minor	Oct. 14, 1893
515	Shute, C. A., Portland House, Eastbourne, Darlington ...	April 11, 1874
516	Sibold, C. W., Public Works Department, Soane Circle, Arrah, Bengal, India	Dec. 9, 1893
517	Simpson, C. L., Engine Works, Grosvenor Road, Pimlico, London...	April 8, 1893
518	Simpson, F. L. G., Mohpani Coal Mines, Gadawarra, C.P., India	A.M. Dec. 13, 1884
		M. Aug. 3, 1889
519	Simpson, F. R., Pelsall Colliery, near Walsall	S. Aug. 4, 1883
		M. Aug. 1, 1891
520	Simpson, J., Heworth Colliery, Felling, R.S.O., Co. Durham	S. Dec. 6, 1866
		M. Aug. 1, 1868
521	Simpson, J. B., Hedgefield House, Blaydon-upon-Tyne (Past-President, *Member of Council)*	Oct. 4, 1860
522	Slinn, T., Plashetts Colliery, Falstone, Northumberland ...	July 2, 1872
523	Smart, A., De Beers Mine, Kimberley, South Africa ...	Feb. 10, 1894
524	Smith, Eustace, Newcastle-upon-Tyne	A.M. June 11, 1887
		M. Aug. 3, 1889
525*	Smith, R. Clifford, Ashford Hall, Bakewell	Dec. 5, 1874
526	Smith, T. E., Phœnix Foundry, Newgate Street, Newcastle-upon-Tyne	Dec. 5, 1874
527	Sopwith, A., Cannock Chase Collieries, near Walsall ...	Aug. 6, 1863
528	Southern, E. O., Ashington Colliery, near Morpeth ...	S. Dec. 5, 1874
		A.M. Aug. 1, 1885
		M. June 8, 1889
529	Southern, R., Burleigh House, The Parade, Tredegarville, Cardiff...	Aug. 3, 1865
530	Southworth, Thos., Hindley Green Collieries, near Wigan	May 2, 1874
531	Spargo, Edmund, 3, Cable Street, Liverpool	June 10, 1893
532	Sparkes, J., Jun., Chester Park, Fishponds, Bristol ...	April 9, 1892
533	Spence, R. F., Backworth, R.S.O., Northumberland ...	S. Nov. 2, 1878
		A.M. Aug. 2, 1884
		M. Aug. 4, 1889
534	Spencer, Francis H., Robinson Gold Mining Co., Witwatersrand, Transvaal	Dec. 13, 1890
535	Spencer, John, Westgate Road, Newcastle-upon-Tyne ..	Dec. 4, 1869
536	Spencer, John W., Newburn, near Newcastle-upon-Tyne...	May 4, 1878
537	Spencer, T., Ryton, Newcastle-upon-Tyne	Dec. 6, 1866
538	Stansfeld, Harold Sinclair, Flockton Manor, Wakefield	A.M. Oct. 13, 1888
		M. Aug. 3, 1889
539	Steavenson, A. L., Durham (Past-President, *Member of Council)*	Dec. 6, 1855
540	Stephenson, G. R., 9, Victoria Chambers, Westminster, London, S.W.	Oct. 4, 1860

		Date of Election and of Transfer.
541	STEVENS, A. J., Uskside Ironworks, Newport, Monmouthshire	April 8, 1893
542	STEVENS, JAMES, 9, Fenchurch Avenue, London, E.C. ...	Feb. 14, 1885
543	STOBART, F., Biddick Hall, Fence Houses	S. Aug. 2, 1873
		A.M. Aug. 5, 1882
		M. June 8, 1889
544	STOBART, H. T., Wearmouth Colliery, Sunderland	S. Oct. 2, 1880
		A.M. Aug. 4, 1888
		M. Aug. 3, 1889
545	STOBART, W., Pepper Arden, Northallerton	July 12, 1872
546	STOBART, WM. RYDER, Etherley Lodge, Darlington ...	Oct. 11, 1890
547	STOBBS, J. T., Walker Colliery, Walker-upon-Tyne ...	Feb. 13, 1892
548	STOKER, ARTHUR P., Birtley, near Chester-le-Street ...	S. Oct. 6, 1877
		A.M. Aug. 1, 1885
		M. Aug. 3, 1889
549	STRAKER, J. H., Howden Dene, Corbridge-upon-Tyne ...	Oct. 3, 1874
550	STREATFIELD, HUGH S., Ryhope Colliery, near Sunderland	A.M. June 8, 1889
		M. Aug. 3, 1889
551	SWALLOW, J., Catchgate, Annfield Plain, R.S.O., Co. Durham	May 2, 1874
552	SWALLOW, R. T., Simonside House, Simonside, South Shields	Feb. 5, 1863
553	SWAN, H. F., North Jesmond, Newcastle-upon-Tyne ...	Sept. 2, 1871
554	SWAN, J. G., Upsall Hall, near Middlesbrough	Sept. 2, 1871
555	SYKES, FRANK K., Springwell Villa, Bishop Auckland ...	S. Feb. 13, 1886
		M. Feb. 10, 1894
556	TATE, SIMON, Trimdon Grange Colliery, Co. Durham *(Member of Council)*	Sept. 11, 1875
557	TAYLOR, HUGH, King Street, Quay, Newcastle-upon-Tyne...	Sept. 5, 1856
558	TAYLOR, T., Quay, Newcastle-upon-Tyne	July 2, 1872
559	TAYLOR-SMITH, THOMAS, Broadwood Park, Lanchester ...	Aug. 2, 1866
560	TEASDALE, THOMAS, Middridge, *via* Heighington, R.S.O. ...	April 9, 1892
561	TELFORD, W. H., Hedley Hope Collieries, Tow Law, R.S.O., Co. Durham	S. Oct. 3, 1874
		A.M. Aug. 6, 1881
		M. June 8, 1889
562	THOMPSON, CHARLES LACY, Farlam Hall, Milton, Carlisle ...	A.M. Feb. 10, 1883
		M. Aug. 3, 1889
563	THOMPSON, R., Jun., 19, The Crescent, Gateshead-upon-Tyne	Sept. 7, 1867
564	THOMPSON, W., Ross Buildings, Charters Towers, North Queensland, Australia	Aug. 4, 1888
565	THOMSON, JOHN, Eston Mines, by Middlesbrough	April 7, 1877
566	THOMSON, JOS. F., Manvers Main Colliery, Rotherham ...	Feb. 6, 1875
567	TINKER, J. J., Hyde, Manchester	April 9, 1892
568	TODD, JOHN T., Bedford Lodge, Bishop Auckland	S. Nov. 4, 1876
		A.M. Aug. 1, 1885
		M. June 8, 1889
569	TONKIN, J. J., Linares, Provincia de Jaen, Spain	Oct. 14, 1893
570	TOUZEAU, E. M., Leadenhall Buildings, Leadenhall Street, London, E.C.	Aug. 6, 1892
571	TRELEASE, W. HENWOOD, Pesterena United G. M. Company, Limited, Pestarena, Vall Anzasca, Novara, Italy ...	April 8, 1893

Date of Election and of Transfer.

572*Tyers, John E., c/o The Hyderabad Deccan Company, Limited, Secunderabad, East India ... ... A.M. Dec. 10, 1877
M. Aug. 3, 1889

573 Tyzack, D., 71, Westgate Road, Newcastle-upon-Tyne ... Feb. 14, 1874

574 Varty, Thomas, Skelton Park Mines, Skelton, R.S.O., Cleveland ... ... Feb. 12, 1887

575 Vaughan, Cedric, Hodbarrow Iron Ore Mines, Millom, Cumberland ... ... Dec. 10, 1892

576 Vitanoff, Geo. N., Sophia, Bulgaria ... ... A.M. April 22, 1882
M. Aug. 3, 1889

577 Vivian, John, Vivian's Boring and Exploration Company, Limited, 42, Lowther Street, Whitehaven ... ... Mar. 3, 1877

578 Waddle, Hugh, Llanmore Iron Works, Llanelly, South Wales ... ... Dec. 13, 1890

579 Wadham, E., Millwood, Dalton-in-Furness ... ... Dec. 7, 1867

580 Wales, H. T., 104, St. Mary Street, Cardiff ... ... Feb. 11, 1893

581 Walker, J. S., Pagefield Iron Works, Wigan, Lancashire Dec. 4, 1869

582 Walker, W., Hawthorns, Saltburn-by-the-Sea ... ... Mar. 5, 1870

583 Walker, Wm., Jun., H.M. Inspector of Mines, Durham ... Aug. 3, 1889

584 Walker, William Edward, Lowther Street, Whitehaven Nov. 19, 1881

585 Wallace, Henry, Trench Hall, Gateshead-upon-Tyne ... Nov. 2, 1872

586 Wallau, Jacob, Messrs. Black, Hawthorn, and Co., Gateshead-upon-Tyne ... ... A.M. Dec. 10, 1887
M. Aug. 3, 1889

587 Walton, J. Coulthard, Writhlington Colliery, Radstock, *via* Bath ... ... S. Nov. 7, 1874
A.M. Aug. 6, 1881
M. June 8, 1889

588 Ward, A. H., Coppull, Chorley ... ... April 14, 1894

589 Ward, H., Rodbaston Hall, near Penkridge, Stafford ... Mar. 6, 1862

590*Ward, T. H., Manager Burma Coal Company, Letkobin, Ihingadaw Circle, Shwebo, Upper Burma, East Indies... A.M. Aug. 5, 1882
M. Aug. 3, 1889

591 Watkyn-Thomas, W., Mineral Office, Cockermouth Castle A.M. Feb. 10, 1883
M. Aug. 3, 1889

592 Watson, Thomas, Trimdon Colliery, Trimdon Grange ... Oct. 11, 1890

593 Webster, H. Ingham, Morton House, Fence Houses ... A.M. April 14, 1883
M. Aug. 3, 1889

594 Weeks, J. G., Bedlington, R.S.O., Northumberland (Vice-President, *Member of Council*)... ... ... Feb. 4, 1865

595 Weeks, R. L., Willington, Co. Durham ... ... A.M. June 10, 1882
M. Aug. 3, 1889

596 Western, C. R., Broadway Chambers, Westminster, London, S.W.... .. .. June 10, 1893

597 White, C. E., Hebburn Colliery, near Newcastle-upon-Tyne S. Nov. 4, 1876
A.M. Aug. 1, 1885
M. Aug. 3, 1889

598 White, H., Walker Colliery, Newcastle-upon-Tyne ... S. Mar. 2, 1867
M. Aug. 5, 1871

599 White, J. F., Wakefield ... ... S. July 2, 1872
M. Aug. 2, 1873

600 White, J. W. H., Bedford House, Roundhay, Leeds ... Sept. 2, 1876

601 Whitehead, Jas., Brindle Lodge, near Preston, Lancashire Dec. 4, 1875

		Date of Election and of Transfer.
602	WHITELAW, JOHN, 118, George Street, Edinburgh	Feb. 5, 1870
603	WHITTEM, THOS. S., Wyken Colliery, near Coventry ...	Dec. 5, 1874
604	WIDDAS, C., North Bitchburn Colliery, Howden, Darlington	Dec. 5, 1868
605	WIGHT, EDWD. S., Hetton Colliery, Carrington, near New-	A.M. Dec. 12, 1885
	castle, New South Wales	M. Aug. 3, 1889
606	WIGHT, W. H., Cowpen Colliery, Blyth	Feb. 3, 1877
607	WILLIAMS, ERNEST, P.O. Box 965, Bittelheim Buildings, Simmonds Street, Johannesburg, Transvaal	Oct. 10, 1891
608	WILSON, J. B., Wingfield Iron Works and Colliery, Alfreton	Nov. 5, 1852
609	WILSON, P. O., c/o Messrs. Howard, Farrar, and Co., P.O. Box 455, Johannesburg, Transvaal	Dec. 9, 1893
610	WILSON, ROBERT, Flimby Colliery, Maryport	Aug. 1, 1874
611	WILSON, W. B., Thornley Colliery, by Trimdon Grange, Co. Durham	Feb. 6, 1869
612	WINSTANLEY, ROBERT, 28, Deansgate, Manchester	Sept. 7, 1878
613	WINTER, T. B., Grey Street, Newcastle-upon-Tyne... ...	Oct. 7, 1871
614	WOOD, C. L., Freeland, Forgandenny, Perthshire	Aug. 3, 1854
615	WOOD, ERNEST SEYMOUR, South Hetton, Sunderland ...	Oct. 10, 1891
616	WOOD, LINDSAY, The Hermitage, Chester-le-Street (PAST-PRESIDENT, *Member of Council)*	Oct. 1, 1857
617	WOOD, THOMAS, North Hetton Colliery Office, Moorsley,	S. Sept. 3, 1870
	Hetton-le-Hole, R.S.O.	M. Aug. 5, 1871
618	WOOD, W. H., Coxhoe Hall, Coxhoe, Co. Durham	Aug. 6, 1857
619	WOOD, W. O., South Hetton, Sunderland *(Member of Council)*	Nov. 7, 1863
620	WOODBURNE, T. J., De Beers Consolidated Mining Company, Limited, Kimberley, South Africa	Feb. 10, 1894
621	WOOLCOCK, J. H., 49, Lowther Street, Whitehaven ...	June 10, 1893
622	WORMALD, C. F., Mayfield Villa, Saltwell, Gateshead-upon-	A.M. Dec. 8, 1883
	Tyne	M. Aug. 3, 1889
623	WRIGHTSON, T., M.P., Stockton-upon-Tees	Sept. 13, 1873
624	YOUNG, JOHN A., 7, Tyne Vale Terrace, Gateshead-upon-	A.M. Dec. 10, 1887
	Tyne	M. Aug. 3, 1889

ASSOCIATE MEMBERS.

Marked * have paid life composition.

		Date of Election.
1	ALLHUSEN, ALFRED, Musgrave House, Gateshead-upon-Tyne ..	Dec. 10, 1892
2	ARMSTRONG, J. H., St. Nicholas' Chambers, Newcastle-upon-Tyne	Aug. 1, 1885
3	ARMSTRONG, T. J., Hawthorn Terrace, Newcastle-upon-Tyne ...	Feb. 10, 1883
4	ATKINSON, G. B., Prudential Assurance Buildings, Mosley Street, Newcastle-upon-Tyne	Nov. 5, 1892
5	AUDUS, T., Mineral Traffic Manager, North Eastern Railway, Newcastle-upon-Tyne	Aug. 7, 1880
6	BELL, B. T. A., Secretary of the General Mining Association of the Province of Quebec, Ottawa, Canada	Aug. 6, 1892
7*	BELL, THOS. HUGH, Middlesbrough-upon-Tees	Dec. 9, 1882
8	BIGGE, EDWARD ELLISON, General Mining Association, Blomfield House, London Wall, London, E.C.	Aug. 1, 1891

		Date of Election
9	Blenkiron, J. Ernest, The Residency, Zomba, British Central Africa	Feb. 11, 1893
10	Blue, Archibald, Bureau of Mines, Toronto, Ontario, Canada...	Feb. 10, 1894
11	Burdon, A. E., Hartford House, Cramlington, Northumberland	Feb. 10, 1883
12	Capell, Rev. G. M., Passenham Rectory, Stony Stratford ...	Oct. 8, 1892
13*	Carr, William Cochran, Benwell Colliery, Newcastle-upon-Tyne	Oct. 11, 1890
14	Cochrane, Henry Heath, Eshwood Hall, Durham	Oct. 10, 1891
15	Cochrane, Ralph D., Hetton Colliery Offices, Fence Houses ...	June 1, 1878
16	Cockburn, W. C., 1, St. Nicholas' Buildings, Newcastle-upon-Tyne	Oct. 8, 1887
17	Cooper, R. W., Newcastle-upon-Tyne	Sept. 4, 1880
18	Davies, Henry, Treharris, R.S.O., South Wales	Dec. 9, 1893
19	Dodds, A. P., 13, Dean Street, Newcastle-upon-Tyne	Nov. 5, 1892
20	Duncan, G. T., 110, Dilston Road, Newcastle-upon-Tyne ..	Aug. 5, 1893
21	East, J. G., 26, Side, Newcastle-upon-Tyne	Feb. 10, 1894
22	Edwards, F. H., Forth House, Bewick Street, Newcastle-upon-Tyne	June 11, 1887
23	Foster, T. J., Coal Exchange, Scranton, Pennsylvania, U.S. ...	Dec. 12, 1891
24	Guthrie, Reginald, Neville Hall, Newcastle-upon-Tyne (Treasurer)	Aug. 4, 1888
25	Hawkins, R. Spencer, P.O. Box 427, Johannesburg, Transvaal	April 14, 1894
26	Hedley, E., Rainham Lodge, The Avenue, Beckenham, Kent ...	Dec. 2, 1858
27	Hedley, J. Hunt, John Street, Sunderland	June 13, 1891
28	Henderson, C. W. C., The Riding, Hexham	Dec. 9, 1882
29	Henzell, Robert, Close, Newcastle-upon-Tyne	April 11, 1891
30	Hicks, James, Redruth, Cornwall	April 14, 1894
31	Hopper, J. I., Tyne Dock, South Shields	Dec. 8, 1888
32	Humphreys-Davies, G., 8, Laurence Pountney Hill, Cannon Street, London, E.C.	Oct. 8, 1892
33	I'Anson, James, Fairfield House, Darlington	Feb. 10, 1894
34	Irvine, Joseph R., Hendon Ropery, Sunderland	Dec. 10, 1887
35	James, Henry M., Colliery Office, Whitehaven	June 10, 1893
36*	Joicey, James John, Sunningdale Park, Berks.	Oct. 10, 1891
37	Krohn, H. A., 103, Cannon Street, London, E.C.	Oct. 14, 1893
38	Palmer, C. B., Usworth Colliery, Washington, R.S.O.	Nov. 5, 1892
39	Patterson, Wm., Front Street, Tynemouth	Aug. 6, 1892
40*	Pease, Arthur, Darlington	Dec. 9, 1882
41	Perkins, Charles, Gallowhill Hall, Newcastle-upon-Tyne ...	Aug. 6, 1892
42*	Proctor, John H., 29, Side, Newcastle-upon-Tyne	June 8, 1889
43	Richter, F., Osborne Villas, Newcastle-upon-Tyne	Oct. 10, 1891
44	Ridley, J. Cartmell, 3, Summerhill Grove, Newcastle-upon-Tyne	Feb. 11, 1893
45	Ridley, Sir Matthew W., Bart., M.P., Blagdon, Northumberland	Feb. 10, 1883

Date of Election.

46 Scurfield, George J., Hurworth-upon-Tees, Darlington ... Dec. 9, 1882
47 Shaw, Saville, Durham College of Science, Newcastle-upon-Tyne ... ... April 13, 1889
48 Snowball, Francis John, Seaton Burn House, Dudley, Northumberland ... ... Dec. 12, 1891

49 Todd, James, Elvet Bridge, Durham... ... ... Aug. 6, 1892
50 Topley, William, 28, Jermyn Street, London, S.W. ... ... Oct. 8, 1889

51 Walton, James, 23, Queen Street, Newcastle-upon-Tyne... ... Aug. 6, 1892
52 Wilson, Arthur P., Mansion House Chambers, Queen Victoria Street, London, E.C. ... ... Aug. 3, 1889
53 Woodward, Harry P., Perth, Western Australia ... ... June 10, 1893

ASSOCIATES.

Date of Election and of Transfer.

1 Archer, William, Victoria Garesfield, Lintz Green ... Aug. 6, 1892

2 Bainbridge, James, North Walbottle Colliery, Newcastle-upon-Tyne ... ... June 10, 1893
3 Barrass, M., Tudhoe Colliery, Spennymoor ... ... S. Feb. 9, 1884
A. Aug. 1, 1891
4 Bates, Sidney, Mickley Colliery Offices, Stocksfield-upon-Tyne ... ... Feb. 8, 1890
5 Bell, John, Wardley Colliery, Newcastle-upon-Tyne ... Feb. 8, 1890
6 Berry, Thomas, 1, Middleton, Swalwell, R.S.O. ... ... Nov. 5, 1892
7 Bolam, J. T., The View, Beamish, Chester-le-Street ... Oct. 14, 1893
8 Bolam, Philip, North Walbottle Colliery, Newcastle-upon-Tyne ... ... Dec. 12, 1891
9 Bowes, Thomas, Pontop House, Annfield Plain, R.S.O. ... Feb. 13, 1892
10 Buglass, J., Stobswood, *via* Acklington, Northumberland... Dec. 10, 1892
11 Burn, Frank H., West Cliff, Elmfield Road, Gosforth, Newcastle-upon-Tyne ... ... S. Feb. 9, 1889
A. Aug. 4, 1894

12 Carr, Robert, Usworth Colliery, Washington, R.S.O. ... Nov. 5, 1892
13 Chandley, Charles ... ... S. Nov. 6, 1880
A. Aug. 3, 1889
14 Clark, Thomas, Dipton Colliery, Lintz Green Station ... Oct. 11, 1890
15 Cockburn, Evan, Page Bank Colliery, *via* Willington, Co. Durham ... ... Aug. 5, 1893
16 Cowx, H. F., Fairfield House, Spennymoor ... ... April 14, 1894
17 Crown, William, 29, Fifth Row, Ashington Colliery, Morpeth ... ... April 14, 1894

18 Dormand, R. B., Waldridge Colliery, Chester-le-Street ... Dec. 9, 1893
19 Draper, William, New Seaham Colliery, Sunderland ... Dec. 14, 1889

20 Emerson, Robert, Tudhoe Colliery, Spennymoor ... ... June 10, 1893

21 Fairs, John, Escomb Bridge, Bishop Auckland ... ... Oct. 14, 1893
22 Fawcett, Edward, Walker Colliery, Walker-upon-Tyne ... June 11, 1892
23 Foster, John William, 24, Silksworth Terrace, New Silksworth, Sunderland ... ... Dec. 14, 1889
24 Fryar, John William, Seghill Colliery, Seghill, Northumberland ... ... June 14, 1890

		Date of Election and of Transfer.
25	GRIFFITHS, ED., Brymbo Colliery, near Wrexham, North Wales	June 13, 1891
26	HALL, JOHN CHARLES, Pegswood Colliery, near Morpeth ...	Dec. 14, 1889
27	HARLE, R. A., Hebburn Colliery, Hebburn, R.S.O. ...	April 14, 1894
28	HENDERSON, WILLIAM, Wheatley Hill Colliery, *via* Trimdon Grange	Aug. 5, 1893
29	HEPBURN, THOMAS, Langley Park, Durham	Feb. 8, 1890
30	HOBSON, MOSES, Shildon, *via* Darlington	Aug. 5, 1893
31	HODGSON, E., Black Boy Colliery, Bishop Auckland ...	Aug. 6, 1892
32	HUNTER, CHRISTOPHER, Broomhill Colliery, Acklington, Northumberland	Dec. 10, 1892
33	JAMES, ALEXANDER A., Croxdale, near Durham	June 10, 1892
34	JOHNSON, WILLIAM, Framwellgate Moor, Durham	Aug. 6, 1892
35	MACKINLAY, EDWARD, East Stanley Colliery, Chester-le-Street	June 11, 1892
36	MACKINTOSH, THOS., Watt's Farm, Blyth	April 11, 1891
37	MASON, BENJ., Burnopfield, R.S.O., Durham	April 11, 1891
38	MINTO, GEORGE WILLIAM, Chapel Row, Ouston, Chester-le-Street	Oct. 10, 1891
39	NICHOLSON, A. D., Lane Ends, Hetton-le-Hole, R.S.O., Co. Durham	S. June 13, 1885 A. Aug. 4, 1894
40	NOBLE, THOMAS GEORGE, Sacriston Colliery, Durham ...	Feb. 13, 1892
41	PARKINSON, W., 6, Ivy Terrace, South Moor, Chester-le-Street	June 13, 1891
42	PATTERSON, THOMAS, Craghead, Chester-le-Street	S. April 13, 1889 A. Aug, 4, 1894
43	PEACOCK, THOMAS, Shelton Collieries, Brook Street, Hanley	April 9, 1892
44	PEDELTY, SIMON, Clara Vale, Ryton-upon-Tyne	Dec. 10, 1892
45	PRINGLE, HY. GEO., Tanfield Lea Colliery, Lintz Green Station, Newcastle-upon-Tyne	S. Dec. 4, 1880 A. Aug. 4, 1894
46	RAMSAY, J. G., Page Bank Colliery, Willington, Durham ...	Dec. 10, 1892
47	RICHARDSON, ROBERT, Blaydon Main Colliery, Blaydon-upon-Tyne	Feb. 8, 1890
48	RIDLEY, GEO. D., Tudhoe Colliery, Spennymoor	Feb. 8, 1890
49	RONTREE, THOMAS, South Boldon Newcastle-upon-Tyne ..	Aug. 2, 1890
50	SCOTT, E. CHARLTON, Rainton Colliery, Fence Houses ...	Oct. 8, 1892
51	SCOTT, WM., Thornhill Collieries, near Dewsbury	S. Mar. 4, 1876 A. Aug. 6, 1892
52	SEVERS, WILLIAM, Beamish View, *via* Chester-le-Street ...	Nov. 5, 1892
53	SHAW, JOHN, Cramlington Colliery, Northumberland ...	April 11, 1891
54	SOUTHERN, JOHN, Heworth Colliery, Newcastle-upon-Tyne	Dec. 14, 1889
55	SPRY, JOHN, Murton Colliery, Durham...	Feb. 11, 1893
56	STANTON, PHILIP, Throckley Colliery, Newcastle-upon-Tyne	S. Dec. 8, 1888 A. Aug. 4, 1894
57	STEAVENSON, C. H., Brotton Mines, Brotton, R.S.O., Yorkshire	S. April 14, 1883 A. Aug. 1, 1891
58	STEPHENSON, WILLIAM, Garesfield Colliery, Lintz Green ...	Dec. 13, 1890
59	SWALLOW, F. C., Simonside House, Simonside, South Shields	April 14 1894

		Date of Election and of Transfer.
60	THOMPSON, JOHN W., Backworth Colliery, Newcastle-upon-Tyne	June 10, 1893
61	THOMPSON, JOSEPH, 38, Church Street, Murton Colliery ...	April 8, 1893
62	WALLETT, T., Heworth Colliery, Felling, R.S.O.	Feb. 10, 1894
63	WALTON, MATTHEW, Dearham Colliery, Carlisle	Dec. 14, 1889
64	WATSON, JOHN, 5, Lovaine Terrace, Low Fell	Dec. 12, 1891
65	WILSON, R. G., Pelton Colliery, Chester-le-Street	Aug. 6, 1892
66	WOOD, JOHN, Coxhoe Hall, Coxhoe, R.S.O., Co. Durham ...	S. June 8, 1889 A. Aug. 4, 1894
67	YEOMAN, THOS., Bengal Coal Company, Limited, Kooldeah Collieries, Giridih, E.I.R., Bengal, India	S. Feb. 14, 1885 A. Aug. 4, 1894
68	ZUMBULOGLON, G. C., No. 3, Astardji han, Grand Bazaar, Constantinople	S. Feb. 14, 1891 A. Aug. 4, 1894

STUDENTS.

		Date of Election.
1	ALLPORT, E. A., Colliery Office, Medomsley, R.S.O., Co. Durham	April 14, 1894
2	BOOTH, F. L., Ashington Colliery, Morpeth	Feb. 10, 1894
3	BROWN, ROBERT O., Elswick Collieries, Newcastle-upon-Tyne ..	Oct. 8, 1892
4	DOUGLAS, DAVID, East Hetton Colliery, Coxhoe, R.S.O., Co. Durham	Aug. 1, 1891
5	KELLETT, MATTHEW H., 5, Elm Street, South Moor, Chester-le-Street	April 11, 1891
6	KING, FRED. Cramlington Colliery, R.S.O., Northumberland ...	Oct. 8, 1892
7	KIRKUP, AUSTIN, Murton Colliery, Sunderland	April 9, 1892
8	KIRKUP, FRED. O., Hutton Henry Colliery, Co. Durham ...	April 9, 1892
9	MASCALL, W. H., c/o Mr. T. Dixon, Adelaide Colliery, near Shildon	Nov. 5, 1892
10	MORDY, WM., Shelton Iron, Steel, and Coal Co., Limited, Stoke-on-Trent	April 11, 1891
11	NOBLE, ALGERNON, Broomhill Colliery, Acklington	Oct. 14, 1893
12	RITSON, JNO. RIDLEY, Burnhope Colliery, Lanchester	April 11, 1891
13	ROBSON, WILLIAM, Jun., Tow Law, R.S.O., Co. Durham ...	June 11, 1892
14	SIMPSON, NELSON ASHBRIDGE, Hedgefield House, Blaydon-upon-Tyne	Aug. 1, 1891
15	SWALLOW, W. A., Bushblades House, Lintz Green	Dec. 9, 1893
16	WALL, GEORGE, The Farm, St. Helens, Bishop Auckland ...	Nov. 5, 1892
17	WATERS, EDWARD, Throckley Colliery, Newcastle-upon-Tyne ...	April 14, 1894
18	WATSON, EDWARD, 19, Bloomfield Terrace, Gateshead-upon-Tyne	Feb. 13, 1892
19	WHITEHEAD, JOHN JAMES, Arden Lea, Heaton, near Bolton ...	Dec. 12, 1891
20	WILLIS, HENRY STEVENSON, Sacriston, Durham	Feb. 13, 1892

SUBSCRIBERS.

1 Owners of Ashington Colliery, Newcastle-upon-Tyne.
2 Birtley Iron Company (3), Birtley.
3 Bridgewater Trustees (3), c/o Clifford Smith, Esq., Bridgewater Offices, Walkden, Bolton-le-Moors, Lancashire.
4 Marquess of Bute, Bute Estate Offices, Aberdare, South Wales.
5 Butterknowle Colliery Company, Darlington.
6 Cowpen Coal Company, Limited (2), F, King Street, Newcastle-upon-Tyne.
7 Earl of Durham (5), Lambton Offices, Fence Houses.
8 Elswick Coal Company, Limited, Newcastle-upon-Tyne.
9 Haswell Coal Company (2), Haswell Colliery, Haswell, *via* Sunderland.
10 Hetton Coal Company (5), Fence Houses.
11 Hutton Henry Coal Company, Limited, 7, Bondgate, Darlington.
12 Marquess of Londonderry (5), c/o V. W. Corbett, Esq., Londonderry Offices, Seaham Harbour.
13 North Brancepeth Coal Company, Limited, Crown Street Chambers, Darlington.
14 Owners of North Hetton Colliery (3), Fence Houses.
15 Ryhope Coal Company (2), Ryhope Colliery, near Sunderland.
16 Owners of Seghill Colliery, Seghill, Northumberland.
17 Owners of South Hetton and Murton Collieries (2), 50, John Street, Sunderland.
18 Owners of Stella Colliery, Hedgefield, Blaydon-upon-Tyne.
19 Owners of Throckley Colliery, Newcastle-upon-Tyne.
20 Owners of Victoria Garesfield Colliery, c/o George Peile, Esq., Shotley Bridge, Co. Durham.
21 Owners of Wearmouth Colliery (2), Sunderland.
22 Westport Coal Company, Limited (2), Manager, Dunedin, New Zealand.

ENUMERATION.

	August 1, 1894.
Honorary Members	26
Members	624
Associate Members	53
Associates	68
Students	20
Subscribers	22
Total	813

Members are desired to communicate all changes of address, or any corrections or omissions in the list of names, to the Secretary.

CHARTER

OF

THE NORTH OF ENGLAND

Institute of Mining and Mechanical Engineers.

FOUNDED 1852.

INCORPORATED NOVEMBER 28TH, 1876.

Victoria, by the Grace of God, of the United Kingdom of Great Britain and Ireland, Queen, Defender of the Faith, TO ALL TO WHOM THESE PRESENTS SHALL COME, GREETING :

WHEREAS it has been represented to us that NICHOLAS WOOD, of Hetton, in the County of Durham, Esquire (since deceased) ; THOMAS EMERSON FORSTER, of Newcastle-upon-Tyne, Esquire (since deceased) ; Sir GEORGE ELLIOT, Baronet (then George Elliot, Esquire), of Houghton Hall, in the said County of Durham, and EDWARD FENWICK BOYD, of Moor House, in the said County of Durham, Esquire, and others of our loving subjects, did, in the year one thousand eight hundred and fifty two, form themselves into a Society, which is known by the name of THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS, having for its objects the Prevention of Accidents in Mines and the advancement of the Sciences of Mining and Engineering generally, of which Society LINDSAY WOOD, of Southill, Chester-le-Street, in the County of Durham, Esquire, is the present President. AND WHEREAS it has been further represented to us that the Society was not constituted for gain, and that neither its projectors nor Members derive nor have derived pecuniary profit from its prosperity ; that it has during its existence of a period of nearly a quarter of a century steadily devoted itself to the preservation of human life and the safer development of mineral property ; that it has contributed substantially and beneficially to the prosperity of the country and the welfare and happiness of the working members of the community ; that the Society has since its establishment diligently pursued its aforesaid

objects, and in so doing has made costly experiments and researches with a view to the saving of life by improvements in the ventilation of mines, by ascertaining the conditions under which the safety lamp may be relied on for security; that the experiments conducted by the Society have related to accidents in mines of every description, and have not been limited to those proceeding from explosions; that the various modes of getting coal, whether by mechanical appliances or otherwise, have received careful and continuous attention, while the improvements in the mode of working and hauling belowground, the machinery employed for preventing the disastrous falls of roof underground, and the prevention of spontaneous combustion in seams of coal as well as in cargoes, and the providing additional security for the miners in ascending and descending the pits, the improvements in the cages used for this purpose, and in the safeguards against what is technically known as "overwinding," have been most successful in lessening the dangers of mining, and in preserving human life; that the Society has held meetings at stated periods, at which the results of the said experiments and researches have been considered and discussed, and has published a series of Transactions filling many volumes, and forming in itself a highly valuable Library of scientific reference, by which the same have been made known to the public, and has formed a Library of Scientific Works and Collections of Models and Apparatus, and that distinguished persons in foreign countries have availed themselves of the facilities afforded by the Society for communicating important scientific and practical discoveries, and thus a useful interchange of valuable information has been effected; that in particular, with regard to ventilation, the experiments and researches of the Society, which have involved much pecuniary outlay and personal labour, and the details of which are recorded in the successive volumes of the Society's Transactions, have led to large and important advances in the practical knowledge of that subject, and that the Society's researches have tended largely to increase the security of life; that the Members of the Society exceed 800 in number, and include a large proportion of the leading Mining Engineers in the United Kingdom. AND WHEREAS in order to secure the property of the Society, and to extend its useful operations, and to give it a more permanent establishment among the Scientific Institutions of Our Kingdom, We have been besought to grant to the said LINDSAY WOOD, and other the present Members of the Society, and to those who shall hereafter become Members thereof, our Royal Charter of Incorporation. NOW KNOW YE that We, being desirous of encouraging a design so laudable and salutary, of our especial grace, certain know-

ledge, and mere motion, have willed, granted, and declared, and do, by these presents, for us, our heirs, and successors, will, grant, and declare, that the said LINDSAY WOOD, and such others of our loving subjects as are now Members of the said Society, and such others as shall from time to time hereafter become Members thereof, according to such Bye-laws as shall be made as hereinafter mentioned, and their successors, shall for ever hereafter be, by virtue of these presents, one body, politic and corporate, by the name of "THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS," and by the name aforesaid shall have perpetual succession and a Common Seal, with full power and authority to alter, vary, break, and renew the same at their discretion, and by the same name to sue and be sued, implead and be impleaded, answer and be answered unto, in every Court of Us, Our heirs and successors, and be for ever able and capable in the law to purchase, acquire, receive, possess, hold, and enjoy to them and their successors any goods and chattels whatsoever, and also be able and capable in the law (notwithstanding the Statutes of Mortmain) to purchase, acquire, possess, hold and enjoy to them and their successors a hall or house, and any such other lands, tenements, or hereditaments whatsoever, as they may deem requisite for the purposes of the Society, the yearly value of which, including the site of the said hall or house, shall not exceed in the whole the sum of three thousand pounds, computing the same respectively at the rack rent which might have been had or gotten for the same respectively at the time of the purchase or acquisition thereof. AND WE DO HEREBY GRANT our especial license and authority unto all and every person and persons and bodies politic and corporate, otherwise competent, to grant, sell, alien, convey or devise in mortmain unto and to the use of the said Society and their successors, any lands, tenements, or hereditaments, not exceeding with the lands, tenements or hereditaments so purchased or previously acquired such annual value as aforesaid, and also any moneys, stocks, securities, and other personal estate to be laid out and disposed of in the purchase of any lands, tenements, or hereditaments not exceeding the like annual value. AND WE FURTHER will, grant, and declare, that the said Society shall have full power and authority, from time to time, to sell, grant, demise, exchange and dispose of absolutely, or by way of mortgage, or otherwise, any of the lands, tenements, hereditaments and possessions, wherein they have any estate or interest, or which they shall acquire as aforesaid, but that no sale, mortgage, or other disposition of any lands, tenements, or hereditaments of the Society shall be made, except with the approbation and concurrence of a General Meeting. AND Our will and

pleasure is, and we further grant and declare that for the better rule and government of the Society, and the direction and management of the concerns thereof, there shall be a Council of the Society, to be appointed from among the Members thereof, and to include the President and the Vice-Presidents, and such other office-bearers or past office-bearers as may be directed by such Bye-laws as hereinafter mentioned, but so that the Council, including all *ex-officio* Members thereof, shall consist of not more than forty or less than twelve Members, and that the Vice-Presidents shall be not more than six or less than two in number. AND WE DO HEREBY FURTHER will and declare that the said LINDSAY WOOD, shall be the first President of the Society, and the persons now being the Vice-Presidents, and the Treasurer and Secretary, shall be the first Vice-Presidents, and the first Treasurer and Secretary, and the persons now being the Members of the Council shall be the first Members of the Council of the Society, and that they respectively shall continue such until the first election shall be made at a General Meeting in pursuance of these presents. AND WE DO HEREBY FURTHER will and declare that, subject to the powers by these presents vested in the General Meetings of the Society, the Council shall have the management of the Society, and of the income and property thereof, including the appointment of officers and servants, the definition of their duties, and the removal of any of such officers and servants, and generally may do all such acts and deeds as they shall deem necessary or fitting to be done, in order to carry into full operation and effect the objects and purposes of the Society, but so always that the same be not inconsistent with, or repugnant to, any of the provisions of this our Charter, or the Laws of our Realm, or any Bye-law of the Society in force for the time being. AND WE DO FURTHER will and declare that at any General Meeting of the Society, it shall be lawful for the Society, subject as hereinafter mentioned, to make such Bye-laws as to them shall seem necessary or proper for the regulation and good government of the Society, and of the Members and affairs thereof, and generally for carrying the objects of the Society into full and complete effect, and particularly (and without its being intended hereby to prejudice the foregoing generality), to make Bye-laws for all or any of the purposes hereinafter mentioned, that is to say: for fixing the number of Vice-Presidents, and the number of Members of which the Council shall consist, and the manner of electing the President and Vice-Presidents, and other Members of the Council, and the period of their continuance in office, and the manner and time of supplying any vacancy therein; and for regulating the times at which General Meetings of the Society and Meetings of the Council shall be held,

and for convening the same and regulating the proceedings thereat, and for regulating the manner of admitting persons to be Members of the Society, and of removing or expelling Members from the Society, and for imposing reasonable fines or penalties for non-performance of any such Bye-laws, or for disobedience thereto, and from time to time to annul, alter, or change any such Bye-laws so always that all Bye-laws to be made as aforesaid be not repugnant to these presents, or to any of the laws of our Realm. AND WE DO FURTHER will and declare that the present Rules and Regulations of the Society, so far as they are not inconsistent with these presents, shall continue in force, and be deemed the Bye-laws of the Society until the same shall be altered by a General Meeting, PROVIDED ALWAYS that the present Rules and Regulations of the Society and any future Bye-laws of the Society so to be made as aforesaid shall have no force or effect whatsoever until the same shall have been approved in writing by our Secretary of State for the Home Department. IN WITNESS WHEREOF WE HAVE CAUSED THESE OUR LETTERS TO BE MADE PATENT.

WITNESS Ourself at Our Palace, at Westminster, this 28th day of November, in the fortieth year of Our Reign.

BY HER MAJESTY'S COMMAND,

CARDEW.

THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

BYE-LAWS.

I.—Constitution.

1.—The North of England Institute of Mining and Mechanical Engineers shall consist of Members, Associate Members, and Honorary Members. The Institute shall in addition comprise Associates and Students.

2.—The Officers of the Institute, other than the Treasurer and the Secretary, shall be elected from the Members and Associate Members, and shall consist of a President, six Vice-Presidents, and eighteen Councillors, who, with the Treasurer and the Secretary (if Members of the Institute) shall constitute the Council.

All Past-Presidents shall be *ex-officio* Members of the Council, so long as they continue Members or Associate Members of the Institute; and Vice-Presidents who have not been re-elected or have become ineligible from having held office for three consecutive years, shall be *ex-officio* Members of the Council for the following year.

II.—Qualifications of Members, Associate Members, Honorary Members, Associates, and Students.

3.—Members.—Every candidate for admission into the class of Members, or for transfer into that class, shall come within the following conditions:—He shall be more than twenty-three years of age, have been regularly educated as a Mining or Mechanical Engineer, or in some other branch of Engineering, according to the usual routine of pupilage, and have had subsequent employment for at least two years in some respon-

sible situation as an Engineer, or if he has not undergone the usual routine of pupilage, he must have been employed or have practised as an Engineer for at least five years. This class shall also comprise every person who was an Ordinary Member, Life Member, or Student on the first of August, 1877.

4.—Associate Members shall be persons connected with or interested in Mining or Engineering, and not practising as Mining or Mechanical Engineers, or in some other branch of Engineering.

5.—Honorary Members shall be persons who have distinguished themselves by their literary or scientific attainments, or who have made important communications to the Society.

6.—Associates shall be persons acting as under-viewers, under-managers, or in other subordinate positions in mines, or employed in analogous positions in other branches of Engineering.

7.—Students shall be persons who are qualifying themselves for the profession of Mining or Mechanical Engineering, or other branch of Engineering, and such persons may continue Students until they attain the age of twenty-five years.

III.—Election and Expulsion of Members.

8.—Any person desirous of becoming a Member, an Associate Member, an Associate, or a Student, shall be proposed according to the Forms A, B, C, or D respectively in the Appendix, in which Form the name, usual residence, and qualifications of the candidate shall be distinctly specified. The Form must be signed by the proposer and at least two other Members or Associate Members, certifying a personal knowledge of the candidate.

Any person qualified to become an Honorary Member shall be proposed according to the Form E in the Appendix, in which Form the names, usual residence, and qualifications of the candidate shall be distinctly stated. This Form must be signed by the proposer and at least five other Members or Associate Members, certifying a personal knowledge of the candidate, and the Council shall have the power of defining the time during which, and the circumstances under which the candidate shall be an Honorary Member.

Any Associate or Student desirous of becoming a Member, shall be proposed and recommended according to the Form F in the Appendix, in which Form the name, usual residence, and qualifications of the candidate shall be distinctly specified. This Form must be signed by the proposer and at least two other Members or Associate Members,

certifying a personal knowledge of the candidate, and the proposal shall then be treated in the manner hereinafter described.

Every proposal shall be delivered to the Secretary, and by him submitted to the next meeting of the Council, who, on approving the qualifications, shall determine if the candidate is to be presented for ballot, and if it is so determined, the Chairman of the Council shall sign such proposal. The same shall be read at the next Ordinary General Meeting, and afterwards be exhibited in the Institute's Hall until the following Ordinary General Meeting, when the candidate shall be balloted for.

A Student may become an Associate at any time after attaining the age of twenty-one years.

9.—The balloting shall be conducted in the following manner :—Each Member or Associate Member attending the meeting at which a ballot is to take place, shall be supplied (on demand) with a list of the names of the persons to be balloted for, according to the Form G in the Appendix, and shall strike out the names of such candidates as he desires shall not be elected, and return the list to the scrutineers appointed by the presiding Chairman for the purpose, and such scrutineers shall examine the lists so returned, and inform the meeting what elections have been made. No candidate shall be elected unless he secures the votes of two-thirds of the Members and Associate Members voting.

10.—Notice of election shall be sent to every person within one week after his election, according to the Form H in the Appendix, enclosing at the same time a copy of Form I, which shall be returned by the person elected, signed, and accompanied with the amount of his annual subscription, or life composition, within two months from the date of such election, which otherwise shall become void.

11.—Every Member having signed a declaration in the Form I, and having likewise made the proper payment, shall receive a certificate of his election, according to the Form J.

12.—Any Member, Associate Member, Associate, or Student elected at any meeting between the Annual Meetings shall be entitled to all Transactions issued in the Institute's year, so soon as he has signed and returned Form I, and paid his subscription.

13.—The Transactions of the Institute shall not be forwarded to those whose subscriptions are in arrear on the 1st of May in each year.

14.—Any person whose subscription is more than one year in arrear shall be reported to the Council, who shall direct application to be made for it, according to the Form K in the Appendix, and in the event of its

continuing one month in arrear after such application, the Council shall have the power, after remonstrance by letter to his last recorded address in the books of the Society, according to the Form L in the Appendix, of declaring that the defaulter has ceased to be a Member.

15.—In case the expulsion of any person shall be judged expedient by ten or more Members or Associate Members, and they think fit to draw up and sign a proposal requiring such expulsion, the same being delivered to the Secretary, shall be by him laid before the next meeting of the Council for consideration. If the Council, after due inquiry, do not find reason to concur in the proposal, no entry thereof shall be made in any minutes, nor shall any public discussion thereon be permitted, unless by requisition signed by one-half of the Members or Associate Members of the Institute ; but if the Council do find good reason for the proposed expulsion, they shall direct the Secretary to address a letter, according to the Form M in the Appendix, to the person proposed to be expelled, advising him to withdraw from the Institute. If that advice be followed, no entry on the minutes or any public discussion on the subject shall be permitted ; but if that advice be not followed, nor an explanation given which is satisfactory to the Council, they shall call a General Meeting for the purpose of deciding on the question of expulsion ; and if a majority of the Members and Associate Members present at such meeting (provided the number so present be not less than forty) vote that such person be expelled, the Chairman of that meeting shall declare the same accordingly, and the Secretary shall communicate the same to the person, according to the Form N in the Appendix.

IV.—Subscriptions.

16.—The annual subscription of each Member and Associate Member shall be £2 2s., of each Associate and Student £1 1s., payable in advance, and shall be considered due on election, and afterwards on the first Saturday in August of each year.

17.—Any Member, Associate Member, Associate or Student may, at any time, compound for all future subscriptions by a payment in accordance with the following scale :—

Under 30	years	of	age,	the sum of £27
Over 30	,,	,,	,,	24
,, 40	,,	,,	,,	21
,, 50	,,	,,	,,	18
,, 60	,,	,,	,,	14

or on such other conditions as the Council may, in writing, accept. Every

person so compounding shall be a Member, Associate Member, Associate or Student for life, as the case may be. Any Associate Member, Associate or Student so compounding who may afterwards be qualified to become a Member, may do so, by election in the manner described in Bye-law 8. All compositions shall be deemed capital money of the Institute.

18.—In case any Member, Associate Member, or Associate who has been long distinguished in his professional career, becomes unable, from ill-health, advanced age, or other sufficient cause, to carry on a lucrative practice, the Council may, on the report of a Sub-Committee appointed by them for that purpose, if they find good reason for the remission of the annual subscription, so remit it. They may also remit any arrears which are due from a Member, or they may accept from him a collection of books, or drawings, or models, or other contributions, in lieu of the composition mentioned in Bye-law 17, and may thereupon release him from any or all future subscriptions, and permit him to resume his former rank in the Institute.

19.—Owners of Collieries, Engineers, Manufacturers, Railway Companies, and Employers of labour generally, may subscribe annually to the funds of the Institute, and each such subscriber of £2 2s. annually shall be entitled to tickets to admit two persons to the rooms, library, meetings, lectures, and public proceedings of the Society; and for every additional £2 2s., subscribed annually, two other persons shall be admissible; and each such subscriber shall also be entitled for each £2 2s. subscription to have a copy of the Transactions of the Institute sent to him.

V.—Election of Officers.

20.—The President, Vice-Presidents, and Councillors shall be elected at the Annual Meeting in August (except in cases of vacancies) and shall be eligible for re-election to any office, with the exception of any President who may have held office for the two immediately preceding years, or Vice-President who may have held office for the three immediately preceding years, and such six Councillors as may have attended the fewest Council Meetings during the past year, and when any such attendances are equal, the Council shall decide between them; but any such Member or Associate Member shall be eligible for re-election after being one year out of office.

Any Retiring Vice-President or Councillor who may be ineligible for re-election shall nevertheless be eligible to any other office.

21.—Each Member, and Associate Member, shall be at liberty to nominate in writing, and send to the Secretary not less than eight days

prior to the Ordinary General Meeting in June, a list, duly signed, of Members and Associate Members suitable to fill the offices of President, Vice-Presidents, and Members of Council, for the ensuing year. The Council shall prepare a list of the persons so nominated, together with the names of the Officers for the current year eligible for re-election, and of such other Members and Associate Members as they deem suitable for the various offices. Such list shall comprise the names of not less than thirty persons. The list so prepared by the Council shall be submitted to the General Meeting in June, and shall be the balloting list for the annual election in August. (See Form O in the Appendix.) A copy of this list shall be posted at least seven days previous to the Annual Meeting, to every Member and Associate Member, who may erase any name or names from the list, and substitute the name or names of any other Member or Associate Member eligible for each respective office; but the number of names on the list, after such erasure or substitution, must not exceed the number to be elected to the respective offices. Papers which do not accord with these directions shall be rejected by the scrutineers. The votes for any Member who may not be elected President or Vice-Presidents shall count for them as Members of the Council, but in no case shall he receive more than one vote from each voter. The Chairman shall appoint four scrutineers, who shall receive the balloting papers, and, after making the necessary scrutiny, destroy the same, and sign and hand to the Chairman a list of the elected Officers. The balloting papers may be returned through the post, addressed to the Secretary, or be handed to him, or to the Chairman of the meeting, so as to be received before the appointment of the scrutineers for the election of Officers.

22.—In case of the decease, expulsion, or resignation of any Officer or Officers, the Council, if they deem it requisite that the vacancy shall be filled up, shall present to the next Ordinary General Meeting a List of persons whom they nominate as suitable for the vacant office or offices, and a new Officer or Officers shall be elected at the first succeeding Ordinary General Meeting.

23.—The Treasurer and the Secretary shall be appointed by the Council, and shall be removable by the Council, subject to appeal to a General Meeting. One and the same person may hold both these offices.

VI.—Duties of the Officers and Council.

24.—The President shall take the chair at all meetings of the Institute, the Council, and Committees, at which he is present (he being *ex-officio* a member of all), and shall regulate and keep order in the proceedings.

25.—In the absence of the President, it shall be the duty of the senior Vice-President present to preside at the meetings of the Institute, to keep order, and to regulate the proceedings. In case of the absence of the President and of all the Vice-Presidents, the meeting may elect any Member of Council, or in case of their absence, any Member or Associate Member present, to take the chair at the Meeting.

26.—At Meetings of the Council, five shall be a quorum. The minutes of the Council's proceedings shall be at all times open to the inspection of the Members and Associate Members.

27.—The Treasurer and the Secretary shall act under the direction and control of the Council, by which body their duties shall from time to time be defined.

28.—The Council may appoint Committees for the purpose of transacting any particular business, or of investigating any specific subject connected with the objects of the Institute. Such Committees shall make a report to the Council, who shall act thereon, and make use thereof as they see occasion.

VII.—Communications and Memoirs.

29.—All papers shall be sent for the approval of the Council at least twenty-one days before a General Meeting, and after approval, shall be read before the Institute. The Council shall also direct whether any paper read before the Institute shall be printed in the Transactions, and immediate notice shall be given to the writer whether it is to be printed or not.

30.—The copyright of all papers communicated to, and accepted for printing by the Council, and printed within twelve months, shall become vested in the Institute, and such communications shall not be published for sale or otherwise, without the written permission of the Council.

31.—Twenty copies of each paper printed by the Institute shall be presented to the author for private use.

32.—All proofs of reports of discussions, forwarded to any person for correction, must be returned to the Secretary within seven days from the date of their receipt, otherwise they will be considered correct and be printed off.

33.—The Institute is not, as a body, responsible for the statements and opinions advanced in the papers which may be read, nor in the discussions which may take place at the meetings of the Institute.

VIII.—Meetings of the Institute.

34.—An Ordinary General Meeting shall be held on the first Saturday of every month (except January and July) at two o'clock, unless otherwise determined by the Council; and the Ordinary General Meeting in the month of August shall be the Annual Meeting, at which a report of the proceedings, and an abstract of the accounts of the previous year, shall be presented by the Council.

35.—All donations to the Institute shall be acknowledged in the Annual Report of the Council.

36.—A Special General Meeting shall be called whenever the Council may think fit, and also on a requisition to the Council, signed by ten or more Members or Associate Members. The business of a Special Meeting shall be confined to that specified in the notice convening it.

37.—The Members, Associate Members, Honorary Members, Associates and Students, shall have notice of, and the privilege to attend, all Ordinary General Meetings and Special Meetings.

38.—Every question, not otherwise provided for, which shall come before any meeting, shall be decided by the votes of the majority of the Members and Associate Members then present.

39.—Invitations shall be forwarded to any person whose presence at the discussions the Council may think advisable, and strangers so invited shall be permitted to take part in the proceedings but not to vote.

Any Member or Associate Member shall have power to introduce two strangers (see Form P) to any General Meeting, but they shall not take part in the proceedings except by permission of the meeting.

IX.—Property of the Institute.

40.—The Funds of the Society shall be deposited in the hands of the Treasurer, and shall be disbursed or invested by him according to the direction of the Council.

41.—The Institute Hall and Reading Room shall be open to the Members, Associate Members, Honorary Members, Associates and Students on every week day, from 10 a.m to 5 p.m., except on such special day or days when the Council shall think it expedient to close the rooms and suspend the circulation of Books. Books shall be issued according to regulations from time to time approved by the Council.

42.—No duplicate copies of any portion of the Transactions shall be issued to any Member, Associate Member, Associate or Student, unless by order from the Council.

X.—Alteration of Bye-laws.

43.—No alteration shall be made in the Bye-laws of the Institute, except at the Annual Meeting, or at a Special Meeting for that purpose, and the particulars of every such alteration shall be announced at a previous Ordinary Meeting, and inserted in its minutes, and shall be exhibited in the room of the Institute fourteen days previous to such Annual or Special Meeting, and such Meeting shall have power to adopt any modification of such proposed alteration of the Bye-laws.

Approved by the Secretary for the Home Department,

Whitehall,
29th June, 1891 and
26th September, 1894.

APPENDIX TO THE BYE-LAWS.

[FORM A.]

A. B. [Christian Name, Surname, Occupation, and Address in full], being upwards of twenty-three years of age, and desirous of becoming a Member of the North of England Institute of Mining and Mechanical Engineers, I recommend him, from *personal knowledge*, as a person in every respect worthy of that distinction, because—

[*Here specify distinctly the qualifications of the Candidate, according to the spirit of Bye-law 3.*]

On the above grounds, I propose him to the Council as a proper person to be admitted a Member.

Signed.....Member or Associate Member.

Dated this.....day of.....18 .

We, the undersigned, concur in the above recommendation, being convinced that A. B. is in every respect a proper person to be admitted a Member.

FROM PERSONAL KNOWLEDGE.

..... } Two Members
..... } or Associate Members.

[*To be filled up by the Council.*]

The Council, having considered the above recommendation, present A. B. to be balloted for as a Member of the North of England Institute of Mining and Mechanical Engineers.

Signed.....Chairman.

Dated this.....day of.....18 .

[FORM B.]

A. B. [Christian Name, Surname, Occupation, and Address in full], being desirous of becoming an Associate Member of the North of England Institute of Mining and Mechanical Engineers, I recommend him, from *personal knowledge*, as a person in every respect worthy of that distinction, and propose him to the Council as a proper person to be admitted an Associate Member.

Signed.....Member or Associate Member.

Dated this.....day of.....18 .

We, the undersigned, concur in the above recommendation, being convinced that A. B. is in every respect a proper person to be admitted an Associate Member.

FROM PERSONAL KNOWLEDGE.

..... } Two Members
..... } or Associate Members.

[*To be filled up by the Council.*]

The Council, having considered the above recommendation, present A. B. to be balloted for as an Associate Member of the North of England Institute of Mining and Mechanical Engineers.

Signed..... ...Chairman.

Dated this.....day of.....18 .

[FORM C.]

A. B. [Christian Name, Surname, Occupation, and Address in full], being desirous of becoming an Associate of the North of England Institute of Mining and Mechanical Engineers, I recommend him, from *personal knowledge*, as a person in every respect worthy of that distinction, and propose him to the Council as a proper person to be admitted as an Associate.

Signed..... .....Member or Associate Member.

Dated this.....day of..... .....18 .

We, the undersigned, concur in the above recommendation, being convinced that A. B. is in every respect a proper person to be admitted as an Associate.

FROM PERSONAL KNOWLEDGE.

... . } Two Members
. ... } or Associate Members.

[*To be filled up by the Council.*]

The Council, having considered the above recommendation, present A. B. to be balloted for as an Associate of the North of England Institute of Mining and Mechanical Engineers.

Signed..... .....Chairman.

Dated this.....day of..... .....18 .

[FORM D.]

A. B. [Christian Name, Surname, Occupation, and Address in full], being under twenty-five years of age and desirous of becoming a Student of the North of England Institute of Mining and Mechanical Engineers, I recommend him, from *personal knowledge*, as a person in every respect worthy of that distinction, and propose him to the Council as a proper person to be admitted as a Student.

Signed.....Member or Associate Member.

Dated this.....day of.....18 .

We, the undersigned, concur in the above recommendation, being convinced that A. B. is in every respect a proper person to be admitted a Student.

FROM PERSONAL KNOWLEDGE.

.... } Two Members
... . } or Associate Members.

[*To be filled up by the Council.*]

The Council, having considered the above recommendation, present A. B. to be balloted for as a Student of the North of England Institute of Mining and Mechanical Engineers.

Signed.....Chairman.

Dated this.....day of..... .....18 .

[FORM E.]

I recommend A. B. [Christian Name, Surname, Occupation, and Address in full], from *personal knowledge*, as a person in every respect qualified to become an Honorary Member of the North of England Institute of Mining and Mechanical Engineers, because—

[*Here specify distinctly the qualifications of the Candidate, according to the spirit of Bye-law 5.*]

On the above grounds, I propose him to the Council as a proper person to be admitted an Honorary Member.

Signed..... .....Member or Associate Member.

Dated this.....day of... ....18 .

We, the undersigned, concur in the above recommendation, being convinced that A. B. is in every respect a proper person to be admitted an Honorary Member.

FROM PERSONAL KNOWLEDGE.

....
.....
....
.. ..
.....

} Five Members or Associate Members.

[*To be filled up by the Council.*]

The Council, having considered the above recommendation, present A. B. to be balloted for as an Honorary Member of the North of England Institute of Mining and Mechanical Engineers.

Signed.....Chairman.

Dated this.....day of... .....18 .

[FORM F.]

A. B. [Christian Name, Surname, Occupation, and Address in full], being at present a.....of the North of England Institute of Mining and Mechanical Engineers, and upwards of twenty-three years of age, and being desirous of becoming a Member of the said Institute, I recommend him, from *personal knowledge*, as a person in every respect worthy of that distinction, because—

[*Here specify distinctly the qualifications of the Candidate, according to the spirit of Bye-law 3.*]

On the above grounds, I propose him to the Council as a proper person to be admitted a Member.

Signed.....Member or Associate Member.

Dated this.....day of.....18 .

We, the undersigned, concur in the above recommendation, being convinced that A. B. is in every respect a proper person to be admitted a Member.

FROM PERSONAL KNOWLEDGE.

..... } Two Members
..... } or Associate Members.

[*To be filled up by the Council.*]

The Council, having considered the above recommendation, present A. B. to be balloted for as a Member of the North of England Institute of Mining and Mechanical Engineers.

Signed.....Chairman.

Dated this..... day of.....18 .

[FORM G.]

List of the names of persons to be balloted for at the General Meeting on....., the.....day of.....18 .

MEMBERS :—

.....

.....

ASSOCIATE MEMBERS :—

.....

.....

HONORARY MEMBERS :—

.....

.....

ASSOCIATES:—

.....

.....

STUDENTS:—

....

....

Strike out the names of such persons as you desire should *not* be elected, and hand the list to the Chairman.

[FORM H.]

SIR,—I beg leave to inform you that on the .....day of..... you were elected a.....of the North of England Institute of Mining and Mechanical Engineers. In conformity with its Rules your election cannot be confirmed until the enclosed form be returned to me with your signature, and until your first annual subscription be paid, the amount of which is £....., or, at your option, a life-composition in accordance with the following scale:—

Under 30 years of age	...	...	...	...	...	£27
Over 30 „	„	...	...	...	...	...	24
„ 40 „	„	...	...	...	...	...	21
„ 50 „	„	...	...	...	...	...	18
„ 60 „	„	...	...	...	...	...	14

If the subscription is not received within two months from the present date, the election will become void under Bye-law 10.

I am, sir, yours faithfully,

.....Secretary.

Dated.....18 .

[FORM I.]

I, the undersigned, being elected a.....of the North of England Institute of Mining and Mechanical Engineers, do hereby agree that I will be governed by the Charter and Bye-laws of the said Institute for the time being; and that I will advance the objects of the Institute as far as shall be in my power, and will not aid in any unauthorised publication of the proceedings, and will attend the meetings thereof as often as I conveniently can; provided that whenever I shall signify in writing to the Secretary that I am desirous of withdrawing my name therefrom, I shall (after the payment of any arrears which may be due by me at that period) cease to be a Member.

Witness my hand this.....day of.....18 .

[FORM J.]

THE NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

Founded 1852. Incorporated by Royal Charter, A.D., 1876.

These are to certify that A. B. [Christian Name, Surname, Occupation, and Address in full] was elected a Member of the North of England Institute of Mining and Mechanical Engineers, at a General Meeting held on the.....day of .....18 .

Witness our hands and Seal this.....day of.....18 .

(Seal)

.....President.

.....Secretary.

[FORM K.]

SIR,—I am directed by the Council of the North of England Institute of Mining and Mechanical Engineers to draw your attention to Bye-law 14, and to remind you that the sum of £.....of your annual subscriptions to the funds of the Institute remains unpaid, and that you are in consequence in arrear of subscription. I am also directed to request that you will cause the same to be paid without further delay, otherwise the Council will be under the necessity of exercising their discretion as to using the power vested in them by the Article above referred to.

I am, sir, yours faithfully,

.....Secretary.

Dated.....18 .

[FORM L.]

SIR,—I am directed by the Council of the North of England Institute of Mining and Mechanical Engineers to inform you, that in consequence of non-payment of your arrears of subscription, and in pursuance of Bye-law 14, the Council have determined that unless payment of the amount £.....is made previous to the.....day of.....next, they will proceed to declare that you have ceased to be a Member of the Institute.

But, notwithstanding this declaration, you will remain liable for payment of the arrears due from you.

I am, sir, yours faithfully,

.....Secretary.

Dated.....18 .

[FORM M.]

SIR,—I am directed by the Council of the North of England Institute of Mining and Mechanical Engineers to inform you that, upon mature consideration of a proposal which has been laid before them relative to you, they feel it their duty to advise you to withdraw from the Institute, or otherwise they will be obliged to act in accordance with Bye-law 15.

I am, sir, yours faithfully,

.....Secretary.

Dated.....18 .

[FORM N.]

SIR,—It is my duty to inform you that, under a resolution passed at a Special General Meeting of the North of England Institute of Mining and Mechanical Engineers, held on the.....day of.....18....., according to the provisions of Bye-law 15, you have ceased to be a.....of the Institute.

I am, sir, yours faithfully,

.....Secretary.

Dated.....18 .

[FORM O.]

BALLOTING LIST,.....

Ballot to take place at the Annual Meeting on.....18.....at Two o'clock p.m.

The names of persons for whom the voter does *not* vote must be erased, and the names of other persons eligible for re-election may be inserted in their place, provided the number remaining on the list does not exceed the number of persons to be elected.

PRESIDENT—Not more than ONE NAME to be returned, or the vote will be lost.

_____ President for the current year eligible for re-election.*

_____ } New Nominations.*

VICE-PRESIDENTS—Not more than SIX NAMES to be returned, or the vote will be lost.

The Votes for any Member who may not be elected President or Vice-Presidents shall count for them as Members of the Council, but in no case shall he receive more than one vote from each voter.

_____ } Vice-Presidents for the current year eligible for re-election.

_____ } New Nominations.

COUNCIL—Not more than EIGHTEEN NAMES to be returned, or the vote will be lost.

_____ } Members of the Council for the current year eligible for re-election.

_____ } New Nominations.

Any list returned with a greater number of Names than ONE PRESIDENT, SIX VICE-PRESIDENTS, or EIGHTEEN COUNCILLORS, Will be rejected by the Scrutineers as informal, and the Votes will consequently be lost.

* To be inserted when necessary.

EX-OFFICIO MEMBERS of the Council for the ensuing year :—

_____ } Past-Presidents.

_____ } Retiring Vice-Presidents.

Extract from Bye-law 21.

Each Member, and Associate Member, shall be at liberty to nominate in writing, and send to the Secretary not less than eight days prior to the Ordinary General Meeting in June, a list, duly signed, of Members and Associate Members suitable to fill the offices of President, Vice-Presidents, and Members of Council, for the ensuing year. The Council shall prepare a list of the persons so nominated, together with the names of the Officers for the current year eligible for re-election, and of such other Members and Associate Members as they deem suitable for the various offices. Such list shall comprise the names of not less than thirty persons. The list so prepared by the Council shall be submitted to the General Meeting in June, and shall be the balloting list for the annual election in August. [See Form O in the Appendix.] A copy of this list shall be posted at least seven days previous to the Annual Meeting, to every Member and Associate Member, who may erase any name or names from the list, and substitute the name or names of any other Member or Associate Member eligible for each respective office ; but the number of names on the list, after such erasure or substitution must not exceed the number to be elected to the respective offices. Papers which do not accord with these directions shall be rejected by the scrutineers. The votes for any Member who may not be elected President or Vice-Presidents shall count for them as Members of the Council, but in no case shall he receive more than one vote from each voter. The Chairman shall appoint four scrutineers, who shall receive the balloting papers, and, after making the necessary scrutiny, destroy the same, and sign and hand to the Chairman a list of the elected Officers. The balloting papers may be returned through the post, addressed to the Secretary, or be handed to him, or to the Chairman of the meeting, so as to be received before the appointment of the scrutineers for the election of Officers.

Names substituted for any of the above are to be written in the blank spaces opposite those they are intended to supersede.

The following Members are ineligible from causes specified in Bye-law 20 :—

AS PRESIDENT .....

AS VICE-PRESIDENT .....

AS COUNCILLORS .....

[FORM P.]

Admit .....

of .....

to the General Meeting on Saturday, the..... .....

(Signature of Member, Associate Member, Associate, or Student).....

The Chair to be taken at Two o'clock p.m.

I undertake to abide by the Regulations of the North of England Institute of Mining and Mechanical Engineers, and not to aid in any unauthorised publication of the Proceedings.

(Signature of Visitor)..... .....

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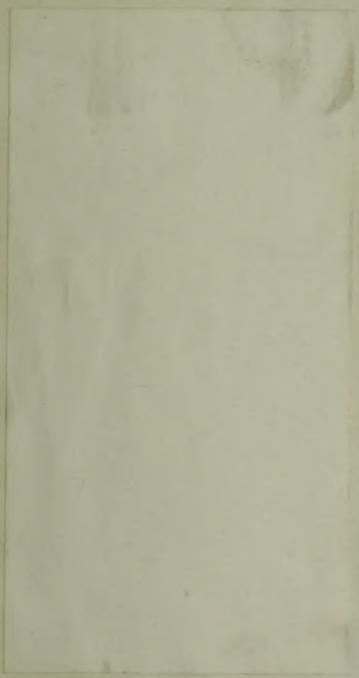

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